MODERN

COPPER SMELTING.

BY

EDWARD DYER PETERS, JR.

TENTH EDITION.

REWRITTEN AND GREATLY ENLARGED.

NEW YORK AND LONDON:
THE SCIENTIFIC PUBLISHING CO.,
1899.

THE AUTHOR
Takes great pleasure in renewing the
DEDICATION OF THIS BOOK
To his Friend
JAMES DOUGLAS,
of New York,
President of the
COPPER QUEEN MINING COMPANY.

PREFACE

TO THE FIRST EDITION.

THE collection of papers which forms this book was mostly prepared in moments stolen from more active professional duties, and must consequently lack the uniformity and completeness which is compatible only with ample leisure and freedom from other more pressing cares.

It has been my intention to confine myself principally to facts gleaned from my own experience, and only to touch upon theoretical questions when essential for the understanding of practical facts.

As the items of cost, both of construction and subsequent operation, are amongst the most important of all the practical questions that face the originators of new smelting enterprises, and as these are virtually unattainable to the general public, I have gone into these figures in considerable detail, not calculating expenses as they appear on paper, and when everything is running smoothly, but giving the actual results of building on a large scale, and of smelting many thousand tons of ores under varying circumstances, and in all of the ordinary kinds of furnaces.

Owing to the magnitude of the subject, I found it impossible to touch upon the so-called "Wet Methods" without increasing the size and consequent cost of this volume to an extent that might probably peril its circulation.

The author desires to acknowledge the valuable assistance of Mr. J. E. Mills, in connection with the geology of the Butte mining district, and to credit Mr. H. M. Howe and Mr. A. F. Wendt with the use he has made of their papers on "Copper Smelting" and on "The Pyrites Deposits of the Alleghanies."

But, above all, he has to thank Mr. James Douglas for a thorough and minute revision and criticism of his manuscript just before publication.

E. D. P., JR.

WALPOLE, MASS., June, 1887.

PREFACE

TO THE SEVENTH EDITION.

SINCE the last thorough revision of this work, the metallurgy of copper has been greatly modified by the success and general introduction of Automatic Calcining Furnaces; by the rapid and extraordinary development of the Copper Bessemer process; by the important and far-reaching improvements in Blast Furnaces and Reverberatories; and perhaps, above all, by the gradual dawning of the idea that, because Copper is worth fifteen times as much as Iron, it is not absolutely necessary to expend fifteen times as much money in handling and treating its ores.

The fusion of sulphide ores by the heat generated from their own oxidation (Pyritic Smelting) has also, lately, become an accomplished fact in several different localities—though how important it is to be as a process pure and simple can scarcely yet be foretold.

The electrolytic refining of pig copper has become such an important feature in many American and European works, that it has been thought proper to include a chapter on this method. I have been most fortunate in securing the assistance of Mr. Maurice Barnett of Philadelphia, whose standing and long practical experience well qualify him to write with authority upon this subject.

All these radical changes have made it necessary to re-write this work, and, also, to add very considerably to its size. Taking advantage of the break caused by a professional trip to Australasia, I have spent months in studying European copper practice, in order to glean what might be useful in our own work. The preparation of this edition, with its numerous elaborate working drawings and plates, has occupied considerably more than a year, and even then would have been impossible without the hearty co-operation of most of the copper smelters of the United States, and the direct assistance of several metallurgists who have written for me on special subjects.

Thus, Mr. Robert Sticht, of Montana, has contributed an exhaustive and valuable paper on Pyritic Smelting. Mr. A. H. Low, of Denver, has furnished much valuable original material in relation to the assaying of copper. Mr. H. A. Keller, Superintendent of the Parrot Silver & Copper Company of Butte, has collaborated with me in writing the chapter on bessemerizing, and has furnished me with material that can only be supplied by a specialist in that department, and that will be appreciated by every copper smelter in this country and in Europe. Mr. I. H. Clutton, of the Messrs. Elliott's Metal Company, Lim., South Wales, has kindly furnished me with a detailed description of the Iodide Copper assay, as practised in Great Britain.

It is impossible for me to even enumerate the names of gentlemen in Europe, Australasia, and America, who have given me assistance in preparing this edition, and in gathering the material for the same. I must, however, particularly thank Mr. Christopher James, of Swansea, Mr. Richard Pearce, of Argo, Colorado, and Mr. C. M. Allen, of Butte, Montana.

I also gratefully acknowledge courtesy and assistance from Messrs. N. P. Hill, W. L. Austin, H. A. Vezin and M. I. Iles, of Denver, Colorado, and wish to thank the Directors of the Boston & Colorado Smelting works, and the Globe Smelter. Also Messrs. E. P. Matthewson, A. S. Dwight, Carl Eilers, and A. Raht, of Pueblo, Colorado, and the Directors of The Pueblo Smelting Company, The Colorado Smelting Company, and The Philadelphia Smelting and Refining Company. Also Mr. Franklin Ballou, of Leadville, Colorado, and the Directors of the LaPlata Smelter and the Arkansas Valley Smelter. Also, Mr. Otto Stalman, of Salt Lake, Utah; and Messrs. W. A. Clark, F. A. Heinze, R. G. Brown, A. H. Wethey, H. Williams, H. C. Bellinger, Captain Palmer, C. W. Parsons and O. Szontágh, of Butte, Montana, and the Directors of The Colorado Smelting & Mining Company, The Parrot Silver & Copper Company, The Butte & Boston Mining & Smelting Company, and The Montana Ore Purchasing Company. Messrs. Frank Klepetko, G. M. Hyams, and the Directors of The Boston & Montana Consolidated Mining & Smelting Company, of Great Falls, Montana. Messrs. H. Thofehrn, V. Ray, and the Directors of The Anaconda Mining Company of

Anaconda, Montana. Mr. A. R. Meyer, President of The Kansas City Consolidated Smelting & Refining Company, of Kansas City, Missouri. The Matthiessen & Hegeler Zinc Company, of LaSalle, Illinois. Mr. H. F. Brown, of Chicago. Mr. Titus Ulke, of Washington. Mr. James Douglas, of New York, President of the Copper Queen Mining Company. Also the late Lord Swansea, and Messrs. T. D. Nicholls and William Terrill, of Swansea, and Mr. Gerard B. Elkington, of Pembrey, Wales. My best thanks are due to the Directors of the Rio Tinto and the Tharsis Companies for their unbounded hospitality at their Spanish mines, and to their local superintendents and other officials. Also to M. Carnot, of the École des Mines, and to MM. Martin and Fenelais, of Paris. Also to the late Professor Stelzner, to Professors Richter and Weisbach, and to the Royal Bureau of Mines at Freiberg. Also to Herren Bergmeister Schroeder, Hüttenmeister Steinbeck, and the Directors of the Mansfelder Gewerkschaft. To the Humboldt Machine Company, of Kalk-on-the-Rhine. Also to Messrs. Bowes Kelly, Wm. Knox, H. H. Sticht, and the Directors of the Broken Hill Mining Company, of New South Wales. To the Hon. John Henry and Mr. G. F. Beardsley, of Tasmania; and many others.

It is only fitting that I should acknowledge my especial indebtedness to Messrs. Fraser & Chalmers, of Chicago, whose knowledge of, and intimate relations with, almost every important mining district in the world, has enabled them to afford me assistance and information that has been of the greatest value.

E. D. P., Jr.

Dorchester, Mass., August, 1895.

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Unless otherwise specified, the "short ton" of 2,000 pounds (907.2 kilos) is used in this work.

One pound (avoirdupois)	0.4536 kilos.
One foot = 12 inches	0.3048 metres.
One gallon	3.7854 litres.
One ounce (Troy weight, as used for precious metals)	31.1 grams.

To reduce "ounces per ton of 2,000 pounds" to per cent., multiply by 0.00343. Example: What is the value, expressed in per cent., of an ore containing 155 ounces silver per ton?

Answer—$155 \times 0.00343 = 0.53165$ per cent.

To change per cent. into ounces per ton of 2,000 pounds, multiply by 292. Example: An ore contains 0.01543 per cent. of gold, how much is this per ton of 2,000 pounds when expressed in ounces?

Answer—$0.01543 \times 292 = 4.5$ ounces per ton.

The common measure of wood (fuel) in the United States is the "cord" of 128 cubic feet.

An American dollar contains one hundred cents, and is equal to about

4 shillings 2 pence	English, or
4 marks 16 pfennige	German, or
5 francs 21 centimes	French.

The value of certain foreign coins in American money has been determined by the United States Treasury Department as follows:

Country.	Title of Coin.	Value in U. S. Money.
Bolivia	Boliviano	$0.72.7
Brazil	Milreis	0.54.6
Chili	Peso	0.91.2
Cuba	Peso	0.92.6
France	Franc	0.19.3
Germany	Mark	0.23.8
Great Britain	Pound	4.86.65
India	Rupee	0.34.6
Italy	Lira	0.19.3
Japan	Yen (gold)	0.99.7
"	Yen (silver)	0.78.4
Mexico	Dollar	0.79
Newfoundland	Dollar	1.01.4
Norway	Crown	0.26.8
Peru	Sol	0.72.7
Portugal	Milreis	1.08
Russia	Rouble	0.58.2
Spain	Peseta	0.19.3
Sweden	Crown	0.26.8

MODERN COPPER SMELTING.

CHAPTER I.

COPPER AND ITS ORES.

COPPER is a red metal, having, when pure and non-porous, a specific gravity of 8.945 in vacuo and at the freezing point of water.*

The slight porosity of ordinary commercial copper reduces this figure to 8.15—8.6.

The capacity of copper for conducting heat stands very high, being 898 when gold is called 1000.

Its electrical conductivity is 931, silver being 1000.

When copper is heated to within 200 or 300 degrees of its melting point, it becomes brittle and friable, and may be actually pulverized.

The melting points of substances that are not easily fusible have not been determined with exactness. As an approximation, copper may be assumed to melt at 2000 degrees Fahr. (1093 degrees Cent.).

When fused it possesses a sea-green color. It is volatile when exposed to our highest attainable temperatures, such as the electric arc, or the oxy-hydrogen flame.

Molten copper has the property of absorbing certain gases, such as hydrogen, carbonic oxide, sulphurous acid, etc., which it sets free again on solidifying. This causes serious difficulties in making copper castings, and requires the employment of special precautions to prevent porosity.

The malleability, ductility, softness, and strength of copper

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* W. Hampe, *Zeitschrift für Berg- Hütten- und Salinenwesen*, 1878, p. 218.

are extraordinarily affected by the presence of minute quantities of certain other substances.

Cuprous oxide dissolves rapidly and homogeneously in metallic copper, and, according to Hampe's researches, produces absolutely no effect until present to the extent of 0.5 per cent. Even 1 per cent. of this substance produces but a very slight diminution in toughness. In greater quantities, it lowers the ductility of the metal, but may be present up to 8 per cent. to 18 per cent. without rendering the copper unfit for most purposes.

Iron, when present, is generally distributed through the copper with much regularity. Reliable data as to the effect of this substance on the tensile strength and electrical conductivity of copper are not to be had. But the weight of evidence seems to show that the minute proportion of iron so frequently present in refined copper has no appreciable effect on any of its useful qualities, except, possibly, its electrical conductivity.

Zinc forms an alloy with copper in every proportion, and slightly lowers its ductility at high temperatures. But until the zinc reaches at least 18 per cent., the tenacity of the alloy at ordinary temperatures does not seem to be lessened.

Lead, while a useful addition to copper that is to be employed for certain mechanical purposes, and assisting in procuring solid castings, has a decidedly injurious effect if present in too large quantities. One-third of one per cent. is sufficient to make the metal both red-short and cold-short, while 0.75 per cent. will ruin copper for any ordinary purpose.

Tin also alloys with copper in all proportions, and begins to injure its ductility when present in quantities of 1 per cent.

Nickel is frequently present in commercial copper, and up to 0.3 per cent. seems to produce no injurious results.

Bismuth is, perhaps, the worst enemy of the copper refiner; for, in spite of its oxidizability, it clings to copper with much tenacity, and affects its properties in the most surprising manner. Hampe finds that merely 0.02 per cent. is sufficient to make the metal distinctly red-short, and cold-shortness begins with 0.05 per cent. bismuth.

Arsenic is not so injurious to copper as is often supposed. Hampe cannot find that 0.5 per cent. of this substance has any bad effect on copper, except to diminish its electrical conductivity; and copper containing 0.8 per cent. arsenic was drawn by him into the finest wire.

Antimony up to 0.5 per cent. acts very much like arsenic, and the lowering of the metal's electrical conductivity does not seem to be accompanied with a loss of either strength or ductility. So Hampe decides after a long series of most careful researches. When present in quantities greater than 0.5 per cent., antimony is much more injurious than arsenic.

Tellurium, which is by no means a rare constituent of copper in certain districts, produces red-shortness, even though present in very small amounts. But Mouchel claims that at ordinary temperatures, the addition of 0.1 per cent. of tellurium largely increases the tensile strength of copper without materially lessening its conductivity.

Silicon has been very thoroughly studied by Hampe as to its effect on copper.*

The addition of 0.5 per cent. of this metal causes a distinct lowering of electrical conductivity. But copper may contain 3 per cent. of silicon before its toughness or malleability is affected. Six per cent. makes copper brittle, increasing as the silicon is increased, until at 11.7 per cent. the alloy is as brittle as glass.

Sulphur in quantities of 0.25 per cent. lowers the malleability of copper. One-half of per cent. produces cold-shortness, though curiously enough such copper is not red-short.

Phosphorus in small quantities seems to produce no injurious effect. At 0.4 per cent. red-shortness is developed.

Carbon is not at all taken up by molten copper, according to Hampe's late researches, though not long ago it was believed to be absorbed in considerable quantities, and to affect the copper most seriously. Too long-continued poling was thought to bring about this result, but it seems to be now conclusively established that the evil effect of "overpoling" copper in the refining furnace is due to the reduction of certain metallic compounds, which, when present in the copper in an oxidized condition, produce no visible effect. Such compounds are arsenates and antimonates of lead and bismuth, mixtures of oxides of lead and copper, etc. The gases arising from poling, such as carbonic oxide, hydrogen, etc., may also be absorbed by the copper, and affect it injuriously.

Tempered copper has been put upon the market by the Eureka Tempered Copper Company, samples of which were examined at the *Versuchsanstalt für Bau-und Maschinen Material* with the following results, the investigation having been made by P. Kirsch:

***Chemiker Zeitung*, 1892, No. 42.

I.—CHEMICAL COMPOSITION.

	Ordinary Copper. Per cent.	Tempered Copper. Per cent.
Silver	0.026	0.025
Copper	99.930	99.981
Tin	....	....
Zinc	....	....
Iron	0.082	0.088
Aluminum	....	....
Arsenic	0.046	0.042
Phosphorus	0.017	0.018
Total	100.101	100.154

As will be seen from the foregoing analyses, the difference of tempered copper from copper of ordinary commercial quality, as far as its composition is concerned, is but slight.

The coppers of which the analyses are given above were mechanically tested, with the following results:

II.—MECHANICAL PROPERTIES.

	Strength in kgs.* per sq. mm.	Elastic Limit in kgs. per sq. mm.	Extension. Per cent.	Contraction in Area. Per cent.
Tension, tempered	18.14	8.05	18.0	26.7
Tension, tempered	19.58	7.67	23.5	36.6
Tension, untempered	16.30	7.13	21.0	36.6
Tension, untempered	17.17	7.08	22.5	35.7
Compression, tempered	39.38	0.42	28.0	....
Compression, tempered	37.20	9.93	26.8	....
Compression, untempered	33.12	9.62	27.4	....
Compression, untempered	36.21	11.20	27.6	....

* 1 kg. per square mm. = 1,425.45 pounds per square inch.

The tests and analyses quoted above were carried out in America, and are quoted for the sake of comparison with those performed at the Versuchsanstalt, which were as follows:

(*a*) *Modulus of Elasticity.*—The modulus of elasticity determined on a specimen tested in tension was 10,000 kilos per square mm. The modulus determined by compression tests was 2,930 kilos per square mm., with a load of 2.5 kilos per square mm., and 1,020 kilos per square mm. with a load of 7.2 kilos per square mm.

(*b*) *Tensile Strength.*—Test pieces used: Sheet, 0.11 mm. in thickness, 50.2 kilos per square mm.; sheet, 0.13 mm. in thickness, 67.9; sheet, 0.55 mm. in thickness, 56.8; sheet, 0.64 mm. in thickness, 53.4; sheet, 1.19 mm. in thickness, 52.3; wire, 0.50 mm. in diameter, 31.8; wire, 0.80 mm. in diameter, 72.0; wire, 1.65 mm. in diameter, 52.0; wire, 2.60 mm. in diameter, 50.0; wire, 4.20 mm. in diameter, 47.6; rod, 87 mm. in diameter, 19.0.

The last named specimen had an elastic limit of 8.1 kilos per square mm. A compression test was made in which deformation began when the load had reached 8.1 kilos per square mm. The load could be increased to 219 kilos per square mm. without producing cracks, although the test piece, which was originally 30 mm. in height, had been shortened to 7.8 mm.

(*c*) *Ductility.*—The extension given by the sheet varied between 0.2—2.0 per cent., while that of the wire was 0.1—0.2 per cent. and that of the rod 13.1 per cent., while the contraction of area at the point of fracture of the latter was 33 per cent. From these tests, as well as by winding tests with the wire, it appears that the material possesses great ductility.

The foregoing series of tests shows that tempered copper possesses properties that distinguish it from the ordinary material, its strength in pieces of small section being noticeably high, although that of larger test pieces is by no means remarkable, as it shows the tensile strength of only 19 kilos per square mm., while ordinary commercial copper gives 20—25 kilos per square mm. Castings made of it are of good quality, and its electrical conductivity is high.*

Copper is easily soluble in nitric acid, in aqua regia, and in strong boiling sulphuric acid.

In dilute sulphuric and muriatic acids, with admission of air, it dissolves slowly.

The metallurgical processes for obtaining copper from the greater proportion of its ores are based upon its strong affinity for sulphur, wherein it exceeds every other metal.

If an impure plate of copper be used as an anode in an acid solution of sulphate of copp r, and another plate be taken as cathode, a properly regulated electric current will precipitate chemically pure copper upon the cathode, the anode dissolving in the same ratio, while the impurities contained in the latter (including often gold and silver), remain as a residual mud. This is

* *Mittheil. Techn. Gewerbe-Museums.* 1891. 261-267, through *Journal of the Society of Chemical Industry* for February, 1892.

the basis of the modern electrolytic refining of copper, first made a technical and commercial success by the Messrs. Elkington of England.

Compounds of Copper Most Important to Metallurgists, and their Reactions.

Copper has two oxides.

Cuprous oxide melts at a red heat without decomposition. It is freely dissolved in molten metallic copper. In a pulverized condition it can easily be changed into the higher oxide by gently heating it in air. It is easily reducible to metallic copper, and forms a fusible slag with silica.

Melted with subsulphide of copper in proper proportions, the copper in both substances is reduced entirely to the metallic state, while the sulphur of the matte combines with the oxygen of the cuprous oxide, and forms sulphurous acid gas. This reaction,and a few analogous ones that follow, are the main basis of our treatment of most ores of copper.

Subjected to the same treatment with sulphide of iron, the copper combines with sulphur to form a subsulphide, which, together with a portion of the undecomposed sulphide of iron, forms a matte, while the iron that has been changed to a protoxide by the oxygen of the cuprous oxide, combines with silica, that should also be present to form slag. The matte being heavier sinks to the bottom and can easily be separated from the supernatant slag, and the first step in the fusion of copper ores has been accomplished.

Cupric oxide is infusible. It is easily reduced to metal by hydrogen or carbonic oxide gases.

It will not combine with silica to form a slag at ordinary metallurgical temperatures unless there be such conditions present that it is mostly reduced to cuprous oxide.

Its reactions with sulphide of iron and silica lead to the same result as in the case of cuprous oxide. That is, we obtain a matte, containing the copper as a subsulphide together with indeterminate sulphides of iron, and a silicate of the ferrous oxide that has been formed, as already explained.

Both of these oxides are soluble in ammonia.

Silicate of copper, in the presence of a strong base, such as ferrous oxide or lime, is reduced by carbon to the metallic state.

When heated with sulphide of iron we obtain sulphide of copper and silicate of iron.

When heated with metallic iron we obtain silicate of iron and metallic copper.

Sulphides of copper.—We know two sulphides of copper: CuS and Cu_2S. Only the latter is of importance metallurgically, as the CuS loses one-half its sulphur at a comparatively low temperature.

This Cu_2S, or subsulphide of copper (cuprous sulphide), melts at a low temperature, and fuses together with sulphide of iron to form an apparently homogeneous substance called matte. This is the object of the first smelting operation with ordinary ores. Much of the sulphide of iron contained in the ore, as well as the sulphides of copper, has been changed into oxides by a preceding calcination at a low temperature. The reactions described above having taken place, we obtain a matte containing all of the copper as a subsulphide with much sulphide of iron, while the alkaline earths and the protoxide of iron combine to a worthless slag.

Under favorable circumstances, and with low-grade ores, we may thus sometimes diminish our material to be treated by 90 per cent. or more in this single fusion, and thus obtain practically all of the copper (as well as any gold or silver present), in a very small quantity of matte, on which we can afford to put considerable expense.

THE ORES OF COPPER.

Although the copper-bearing minerals are numerous, yet those of commercial importance are few in number, and, for the most part, quite simple in chemical composition. The following minerals may be properly considered ores of copper, and are found in the United States in the localities enumerated.

NATIVE METALLIC COPPER.

Aside from the extensive occurrence of this metal in the Lake Superior region and, more sparingly, at Santa Rita, New Mexico, it is found very frequently as a product of decomposition, though seldom in sufficient quantities to render it of any commercial importance. It is usually remarkable for its purity.

CUPRITE, OR RED OXIDE OF COPPER, Cu_2O; 88.8 Cu, 11.2 O.

This mineral occurs solely as a product of decomposition, and while quite widely distributed, is nowhere an ore of any importance, except in the Southwestern carbonate mines, where it sometimes permeates large masses of iron oxide, notably increasing their copper contents. Quite large lumps of this mineral are found in the Santa Rita mines, and are evidently the result of an oxidation of nodules of metallic copper, the unaltered center being usually preserved of greater or less size.* Many of the Butte City veins, as well as fissures throughout the Eastern Coast Range, carry this mineral in their upper portions as a product of the decomposition of sulphide ores.

MELACONITE, BLACK OXIDE OF COPPER, CuO; 79.8 Cu, 20.2 O.

This ore, with its metallic contents usually in part replaced by oxides of iron and manganese, is not quite so widely distributed as the sub-oxide, but is more frequently found in masses sufficiently large to pay for extraction. Its most remarkable occurrence in the United States was in the Blue Ridge mines of Tennessee, North Carolina, and Virginia, where the upper portion of the beds furnished a very large amount of from 20 to 50 per cent. ore, having the appearance of melaconite, and giving rise to expectations that were always shattered after passing through this rich zone and reaching the lean, unaltered pyrites below. This so-called black oxide of the Blue Ridge region† seems to be an inti-

*An average sample of thirteen tons of concentrates, taken by the author at Santa Rita, in 1881, and partially analyzed under his supervision, gave, after continuing the concentration by hand to almost complete removal of the rock constituents :

Oxides of copper	13.42
Carbonates of copper	1.27
Oxides of iron	0.13
Metallic iron (from stamps)	0.29
Sulphur	0.11
Insoluble residue	0.37
Metallic copper	83.66
Zn, Ag, Co, Ni, Pb, Mn	Traces
	99.25

This analysis presented points of considerable difficulty, especially in determining the amount of oxide of copper in the presence of metallic copper. Entirely satisfactory results were not obtained ; but the method proposed by W. Hampe, by means of nitrate of silver, yielded the only figures that could lay the slightest claims to accuracy. Since this was written, the combustion method has been so perfected that it will undoubtedly take the place of Hampe's process in determining the amount of oxygen in the presence of metallic copper.

† *Pyrites Deposits of the Alleghanies*, by A. F. Wendt.

mate mixture of glance, oxide, carbonate, and sometimes finely divided native copper. Two analyses, by Dr. A. Trippel, show their constituents:

Oxide of copper	5.75	3.80
Sesquioxide of iron	1.50	63
Sulphur	18.75	25.40
Copper	71.91	41.00
Iron	.93	26.56
Soluble sulphates of copper and iron	.72	1.78
	99.56	99.17

A pile of such ore, laid on a bed of cordwood and moistened, often ignites the wood below, and thus roasts itself without firing.

MALACHITE, $CuCO_3 + Cu(OH)_2$; 71.9 CuO, 19.9 CO_2, 8.2 HO.

This is a much more valuable compound of copper than the two preceding oxides, from a commercial standpoint; although no mines in the United States furnish malachite of sufficient purity to fit it for ornamental purposes.

While it may be said to occur in widely distributed but ordinarily in non-paying quantities, in the upper decomposed regions of most copper deposits, there are certain localities in which it forms the principal ore of this metal. It is very seldom found in a state of purity, but is mixed with various salts of lime and magnesia, oxides of iron and manganese, silica in its various forms of quartz, chalcedony, flint, chert, and jasper, and, when seemingly present in large quantities, it often forms only worthless incrustations, or merely colors nodules and masses of valueless material. It is then difficult, and in some cases impossible, to form any accurate opinion of the tenor of the ore from its external appearance. It is an important ore in the Southwest.

AZURITE, $2\,CuCO_3 + Cu(OH)_2$; 69.2 CuO, 25.6 CO_2, 5.2 HO.

This mineral requires only a passing notice. It is distributed in the same manner and occurs under the same conditions as its sister carbonate, but in very much smaller amounts. It occurs in profitable quantities only in some of the Southwestern mines. Specimens of this mineral are found with malachite and calc-spar in the Longfellow mine, exceeding in beauty anything of the kind that is known elsewhere in the United States.

CHALCOPYRITE, Cu_2S, Fe_2S_3; 34.4 Cu, 30.5 Fe, 35.1 S.

This is by far the most widely distributed ore of this metal, and furnishes the greater proportion of the world's copper. It occurs principally in the older crystalline rocks, frequently accompanied with an overwhelming percentage of iron pyrites, in bedded veins, in Newfoundland, in Quebec, Canada, in Vermont, Virginia, Georgia, Tennessee, and Alabama.

The value of copper-bearing fissure veins below the limit of surface decomposition is nearly always due to this mineral. In some localities the chalcopyrite forms with pyrite a fine-grained mechanical mixture, varying in color with its percentage of copper from deep yellow to steel-gray. This substance is easily recognized under the microscope as a mechanical mixture, and not a chemical compound. In most of the carbonate mines of the Southwest that have attained any considerable depth, chalcopyrite is already becoming apparent, in minute specks; and it is highly probable that the altered ores near the surface, with their valuable admixture of ferric oxides, are all due to the decomposition of this mineral. The sulphureted fissure-veins of the Rocky Mountains and Sierra Nevada are seldom free from this mineral, although their value almost invariably depends upon their precious metal contents. The remarkable purple ores and copper glance of Butte City, Montana, have already in several mines given place in depth to the universal yellow sulphide.

In the vast bodies of bisulphide of iron (iron pyrites), that furnish so large a proportion of the material for the world's manufacture of sulphuric acid, copper pyrites is frequently present in paying quantities.

Silver and gold are also commonly present in minute amounts, and one of the most interesting feats of metallurgical chemistry is the profitable extraction of these metals from ores carrying only 3 per cent. copper, less than one ounce of silver, and four or five grains of gold. (It is a curious fact that monosulphide of iron, though often rich in copper pyrites, and noticeably so in nickel, very rarely carries more than the merest traces of the precious metals.)

In these great pyrites deposits, a concentration of the silver frequently takes place just at the line of junction between the oxidized surface gossan and the unaltered pyrites below.

At one of the surface openings on the Rio Tinto deposit, the line of demarcation between the oxidized and sulphide ores is very sharp. The gossan at this place was about sixty feet thick, and just below this capping of red iron ore, and resting upon the almost unaltered pyrites, was a soft, grayish earthy deposit from half an inch to six inches in thickness, and containing from 50 to 150 ounces silver (0.17 per cent. to 0.52 per cent.) per ton, while the original pyrites contained probably about one ounce, or less, to the ton. A sample that I took from many spots assayed 66 ounces (0.23 per cent.) silver and 9 per cent. lead.

I had the pleasure of witnessing the discovery and development of a still more striking instance of this nature at the Mount Lyell mine in Tasmania. This is a deposit of massive iron pyrites about 300 feet in width, and averaging 4½ to 5 per cent. copper, with 3 pennyweights gold and 3 ounces silver per ton (0.005 per cent. gold and 0.01 per cent. silver). At one portion of the deposit there is an extensive and remarkable chute of gossan on the footwall, descending to the 200-foot level. This gossan contains nearly the same amount of gold as the original pyrites, from which it was doubtless derived, but neither silver nor copper. These commercially important constituents had been leached out and redeposited on the footwall, under the lower border of the gossan, in a series of extensive and irregular pockets, which are still being actively worked. The first 50 tons of ore extracted averaged very close to 2,000 ounces silver per ton (nearly 7 per cent.) and 21 per cent. copper. The ore was a pure chalcopyrite, containing streaks and nodules of a very rich copper-silver glance.

In our own country, the United Verde mine in Arizona has a layer, very rich in silver, between the gossan and the ordinary pyrites.

CHALCOCITE, COPPER GLANCE, Cu_2S; 79.7 CU, 20.3 S.

This ore is seldom found in a condition of perfect purity, its valuable component being frequently in part replaced by iron and other metals. Its copper percentage rarely falls below 55, and even at this low standard the mineral retains its physical characteristics, a slight diminution in its luster being the principal difference observable. When high in copper, it greatly resembles the white metal of the smelter. Chalcocite containing from 60 to 74 per cent. of copper occurs in large amounts in the noted Anaconda mine, Butte, Montana. Several of the other Butte mines carry the same mineral, although, as they approach the western boundaries of the district, it gradually passes into bornite or peacock ore. It is also an important ore in Arizona, occurring in large quantities near Prescott, as well as in the Coronado and other Clifton mines. In New Mexico, it constitutes virtually the entire value of the Nacimiento and Oscura Permian beds. It occurs frequently in Texas in the Grand Belt mines, and is the principal ore of numerous narrow fissures in the Middle and Atlantic States. In the Orange Mountains of New Jersey, examined by the author, it was found in a species of shale, as an ore of the following composition:

Copper	75.20	Sulphur	17.97
Iron	4.10	Insoluble	1.10
Manganese	1.13		
Silver (2.37 ounces)	0.01		99.51
Gold	Trace		

BORNITE OR ERUBESCITE, $3\ Cu_2S,\ Fe_2S_3$; 55.58 CU, 16.36 FE, 28.06 S.

This is one of the most beautiful of the sulphureted ores of copper, being characterized in its fresh condition by a superb purplish-brown color, which soon changes on exposure to the air into every conceivable hue, from a golden yellow to the deepest indigo, and from a brilliant green to a royal purple. The mode of occurrence of this mineral and its limited extent of distribution as regards depth indubitably stamp it as a product of decomposition, solution, and re-deposition of the metallic portion of the vein. Like copper glance, this mineral is far from uniform in its composition, varying in richness from 42 to nearly 70 per cent. of copper without entirely losing its characteristic colors.

It forms a frequent ore of the Butte mines in their rich zone, which lies between the leached-out surface zone and the unaltered sulphides below.

TETRAHEDRITE, GRAY COPPER ORE, FAHLORE $(Cu_2S,\ FeS,\ ZnS,\ AgS,\ PbS)_4\ (Sb_2S_3,\ As_2S_3)$; 30.40 PER CENT. COPPER.

Except in those rare and highly argentiferous varieties in which the copper is replaced to a greater or less extent by silver, this is seldom regarded in the United States as an ore of copper.

Both its scarcity and its obnoxious components (arsenic, antimony, etc.) prevent it use as a source of copper in this country, where the general purity of our ores has established such a high standard for this metal. Only the most favorable circumstances, mineralogical, metallurgical, and commercial, would render the working of non-argentiferous fahlores at all practicable. This mineral occurs in small quantities in certain of the Butte copper mines, rendering their product slightly inferior to that from the oxidized ores of Arizona or the pure sulphides of Vermont. This slight disadvantage is, however, far outweighed by their contents in silver, which partly owes its presence to this same arsenical mineral. From the San Juan region, Colorado, an argentiferous tetrahedrite adds a notable quantity to the production of the United States. It appears principally as matte from the lead furnaces, and as black oxide from the Argo separating works.

CHAPTER II.

DISTRIBUTION OF THE ORES OF COPPER.

THE ores of copper are widely distributed over the earth's surface, and may be found in almost every geological formation. The principal copper districts of North America may be classed in three groups:

I. The Atlantic coast beds.
II. The Lake Superior deposits.
III. The deposits of the Rocky Mountains and Sierra Nevadas.

I.—THE ATLANTIC COAST BEDS.

Throughout its whole extent in North America, the Atlantic coast is bordered by a succession of parallel ranges, which, by their general geological as well as geographical analogy, must be classed in the same system. They form an unbroken chain from Florida to Labrador, and thence, continuing their same northeasterly direction along the coast of that bleak country, dip beneath the waters of Baffin's Bay, where they are represented by a series of submarine peaks, and, nourishing the gigantic glacier system of western Greenland,* terminate, so far as known, in Mount Edward Parry, north latitude 82 degrees 40 minutes. Dr. T. Sterry Hunt's admirable researches have given us a very clear insight into the origin, formation, and structure of this immense range of mountains within the confines of the United States and Canada. It consists essentially of metamorphic rocks—largely crystalline schists—and is metal-bearing to a greater or less degree throughout its entire extent, though only in a few places is copper found in a sufficiently concentrated form to justify any attempts at extraction.

The main copper mineral of importance in this range is chalcopyrite. In the more northerly division, where there has been extensive glacial denudation, this reaches unaltered almost, or quite, to the grass-roots, while from Virginia to Tennessee, where

* See Dr. Kane's *Arctic Expedition* for soundings taken in Baffin's Bay; also *Geology of Greenland's Mountains.*

abrasion has not taken place, and where oxidation has been assisted by climatic influences, decomposition with subsequent concentration is found to a considerable depth. The result of this is usually a zone, rich in an impure black oxide of copper containing a certain proportion of sulphur, which sometimes occurs in considerable quantities near the surface, after first passing through a greater or less extent of barren iron oxide, derived from pyrite, and which has no doubt furnished the copper to enrich the underlying zone.

The occurrence of this valuable mineral in merchantable quantities has, in more than one instance, raised expectations and led to large expenditures that have subsequently proved entirely unwarranted; for at a slightly greater depth, the unaltered vein assumes its true character of a more or less solid pyrite or pyrrhotite carrying a very small amount of copper (seldom above 3 per cent.) in the form of the common yellow sulphuret. When the accompanying mineral is a bisulphide of iron and the locality is favorable, the pyrite may be utilized in the manufacture of sulphuric acid, the copper being extracted from the residues by well-known methods; but when the prevailing mineral is the *mono*sulphide—magnetic pyrites—there can be no question of profitable working, pyrrhotite being absolutely valueless since copperas has become a by-product of fence-wire making. At Capelton, in Canada, at Ely, Vermont, and at one or two points in Newfoundland, copper pyrite occurs in a sufficiently concentrated form to yield from 5 to 6 per cent. in considerable quantities, an ore on which profitable operations may be conducted, under favorable conditions.

In Virginia, at Ore Knob, North Carolina, at the Tallapoosa mine in Georgia, and at Stone Hill, Alabama, indications of a similar concentration of copper have given rise to extensive explorations, and, in some cases, to the expenditure of large amounts of money, which have not always resulted satisfactorily. These are all examples of so-called *bedded veins*, following the lines of stratification, and being simply sandwiched in between the layers of rock. One of the most curious features of these beds is the alternate occurrence of the sulphide of iron that forms the great mass of the gangue, as pyrrhotite and pyrite. In Capelton, for instance, we have the bisulphide; a hundred miles distant, at Ely, the monosulphide alone exists; in Virginia and at Ore Knob, the monosulphide preponderates; while in the Tallapoosa mine, the bisulphide alone is found. Neither the chemical nor geological composition of the corresponding country-rock explains this phenomenon. Here, it will be proper to mention the occurrence, in stratified rocks, of the sulphide of copper (copper glance), usually in unimportant quantity, throughout Pennsylvania, New Jersey, and other Middle and Southern States.

For convenience, we may append to this division the copper ores of Sudbury, in the Province of Ontario, Canada. Copper pyrites is here associated with much greater amounts of nickeliferous pyrrhotite, and occurs in stockwerks in the Huronian rocks, along or near contacts of diorite and gneiss, or diorite and quartz-syenite.

II.—THE LAKE SUPERIOR DEPOSITS.

These deposits occur in the Keweenawan series, which are upturned rocks of Algonkian age that have been deposited unconformably upon the iron-bearing Huronian series, and are in turn overlapped by the sandstones of the Cambrian.

The copper-bearing strata of the Keweenawan series consist of beds of trap, sandstone, and conglomerates of doubtful age. They rise at an angle of 45 degrees out of the horizontal sandstone from which the basin of Lake Superior has been eroded. It is only on the Keweenaw promontory of Michigan that they have yielded copper in profitable amounts, though the same series of rocks, always containing a certain proportion of this metal, stretches westward across Wisconsin, far into Minnesota. In the latter State, sulphurets of copper are sometimes present in the Keweenaw belt, but in Michigan the copper occurs exclusively in the metallic condition, and is believed to be derived from the solutions formed from the oxidation of the cupriferous sulphides that abound in the underlying Huronian formation.

Three classes of deposits have been exploited on the Keweenaw peninsula.

1. Veins which in some instances cut, and in others are parallel with the beds, but which are filled with vein stone different from the intersected rocks. It is from these veins that the great masses of native copper have been derived that have made such an impression upon the public.

2. Copper-bearing beds of amygdaloidal diabase, locally called *ash beds*, and amygdaloidal traps.

3. Beds of conglomerate, of which the cementing material consists in part of copper.*

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* "The Copper Resources of the United States," by James Douglas, *Journal of the Society of Arts*, London.

The mass mines become poorer in depth, and are considered somewhat hazardous enterprises.

The *ash beds* have been much more profitable, the Quincy, Franklin, Pewabic, and many other noted mines, belonging to this class.

The conglomerate beds produced in 1893, 85,662,000 pounds of fine copper, or 75 per cent. of the entire output of the Lake Superior District. Yet this large production comes from a single ore-chute, about three miles in length, and penetrated to nearly 4,000 feet in depth. On this chute are situated the Calumet & Hecla, Tamarack,and Atlantic Mines.

The average contents of the ores now mined at Lake Superior may be placed at about 2.9 per cent.

III.—THE DEPOSITS OF THE ROCKY MOUNTAINS AND THE SIERRA NEVADAS.

This division includes a heterogeneous collection of districts and formations, and comprises nearly one-half the area of the United States.

The rock formations of the different mining regions in this district happen to differ sufficiently to enable us to subdivide it according to its geological characteristics.* We have:

(*A*) *Precambrian deposits*, which include the celebrated mines of Butte, Montana, and probably the United Verde group of Prescott, Arizona.

(*B*) *Palæozoic deposits*, always associated with eruptive rocks in the profitable mines of this country. The most important areas of this class are:

(*a*) Bisbee, Clifton, and Globe, in Arizona, in which the ore occurs mainly in the lower carboniferous limestones, and is ordinarily oxidized to a depth of some hundreds of feet.

(*b*) Leadville, Colorado, the copper sulphides being found in conjunction with pyritous silver ores in silurian limestones in contact,and fracture-planes.

(*c*) Tintic, Utah, in Palæozoic limestones, with ores of gold and silver.

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* "The Geological Distribution of the Useful Metals in the United States." See paper by S. F. Emmons, *Transactions American Institute Mining Engineers*, Vol. XXII., p. 53, from which is also taken certain information regarding the geology of the Lake Superior district.

(*C*) *Mesozoic Deposits.*—The most important of these deposits, from the commercial standpoint, are the pyritous beds of California, that occur along the foothills of the Sierra Nevada, at the contact of diabase and the upturned cretaceous slates. In Texas and Colorado, and especially in New Mexico, there are areas of Trias that show a wide distribution of disseminated copper, and some few points of sufficient concentration to warrant exploitation. In the cretaceous throughout the Rocky Mountains, copper occurs to a subordinate degree, in connection with ores of the precious metals, whence there is a considerable production of the less valuable metal, as a by-product.

(*D*) *Tertiary and Recent Deposits.*—No copper worth mentioning is produced in the United States from such rocks. There are, however, several spots in Arizona and New Mexico, where there are recent deposits resulting from the surface leaching of copper minerals situated in the older rocks.

*The Butte Mines.**—The most northerly, and by far the most important of all the ores included in this divison, are the deposits of Butte, Montana, their output for 1893 being 155,000,000 pounds of fine copper, or about 69,200 tons of 2,240 pounds. This enormous production came from a little granitic area of (probably) precambrian rocks, not over one mile wide by two miles long, situated on the western slope of the main divide of the Rocky Mountains.

The ore occurs in irregular lodes in the granite, having an east and west strike, and an average dip of some 12 degrees to the south from the vertical, though in places this becomes as much as 45 degrees. The distribution of the ore is also very irregular, extensive bodies of the same being frequently found on breaking through what appears to be a well-defined wall. Again, there will be no definite line between the vein and the adjacent country rock. The ore is usually found in chutes that often extend for several hundred feet along the strike, before pinching out. Their depth is frequently even greater than their length, though they are sometimes broken by small faults.

* For a detailed description of the mines and metallurgical works of this district, see a paper by the author, entitled "The Mines and Reduction Works of Butte City, Montana," *United States Geological Survey, Mineral Resources*, Albert Williams, Jr., 1885. For a recent and more valuable description of the mines of Butte, see "The Ore Deposits of Butte City," by R. G. Brown, *American Institute Mining Engineers*, October, 1894. I have used this paper freely in the present section.

The veins are rarely banded, and vary in size from a few inches of compact ore up to 100 feet or more, as in the Anaconda mine. Five or six feet may be regarded as the ordinary width, though a large proportion of the ore raised in Butte comes from stopes of much greater width than this. The gangue rock is usually granitic and silicious, but not quartzose.

The croppings of the copper veins are moderately prominent, and consist of the usual brownish, iron-stained quartz that may be found at almost any point in the great American mountain chain, from Alaska to Patagonia. Just below the surface, red and yellow oxides of iron appear, carrying high values in silver and gold, but usually low in copper.

These decomposed ores extend to the water level, which is reached at a depth of from 40 to 300 feet, depending upon the surface irregularities. At this point begins the zone of rich, secondary copper ores that have made Butte so famous. The copper minerals of this zone are difficult to determine, as they pass through all gradations from pure chalcocite down to chalcopyrite, bornite being also of very frequent occurrence. Iron pyrites is usually present in considerable amounts.

Naturally, these rich, secondary ores have fallen off in depth, Ledoux estimating their average decline at 2 per cent. copper per 100 feet. But this diminution lessens as greater depth is gained, and the ore raised from the Butte mines at present, omitting a few bonanza bodies, averages about 6½ per cent. copper and 5 ounces silver to the ton of 2,000 pounds (0.017 per cent. silver). There is a loss of some 18 per cent. in concentration, and to this must be added the smelting loss, which will reduce the yield of the great bulk of the Butte ores to 5 per cent. copper and 4 ounces (0.014 per cent.) silver.

Notwithstanding this great decline in percentage and values (which, to be sure, has resulted partly from the ability to work lower grade ores to advantage), the Butte mines are making more profit to-day from a 6 per cent. ore than formerly from one of double this richness. This results from the consolidation of mining properties, and from the astounding and radical improvements made in the metallurgical treatment of the ore. The rich surface ores of the district have furnished the capital that was needed to design and construct the improved plants, and to gain experience necessary for treating the lower grade ores at a profit.

Ledoux* makes the following four statements, with which I agree in the main:

1. The average yield of copper in the Butte camp is 5½ per cent., or 110 pounds per ton net, and it will not fall much below 5 per cent.

2. This copper costs 9½ cents per pound, delivered in New York.

3. The value of the precious metals in the copper is equal to $57 per ton copper, and should yield a net profit of over 2 cents per pound of copper, with silver at 65 cents per ounce, and electrolytic copper at 9½ cents per pound. (This was written a year ago.)

4. The present output can be maintained for at least ten years to come.

At the present low price of the metals in question, it may be assumed that the net profits of the Butte mines are mainly derived from their silver contents, the copper just about paying all the expenses. Aside from the Calumet & Hecla ore chute at Lake Superior, this is probably doing better than any other great copper district in the world.

At a depth exceeding 1,300 feet, there is no sign of any weakening or giving out of the Butte copper lodes.

The Arizona Copper Mines.†—These comprise four distinct groups of deposits of commercial importance, besides a very large number of slightly developed districts, some few of which may yet become producers. The production in 1893 was about 44,000,000 pounds, or 19,643 tons of 2,240 pounds.

The profitable mines have been found mostly in carboniferous limestone, and at, or near, its contact with an eruptive rock, such as felsite, diorite, or porphyry. On entering an underlying acid rock, whether sandstone or porphyry, the veins become narrow and unprofitable. The productiveness and permanency of most of the Arizona copper districts seem to stand in close relation to the thickness of the ore-bearing limestone. A striking example of this fact may be seen in the accompanying cut, Fig. 1, which shows a section across the well-known Longfellow mine of Clifton, Arizona.

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* *The Mineral Industry*, Vol. II., p. 245.

† The cuts and many of the facts in this description are from A. F. Wendt's paper, "The Copper-Ores of the Southwest," *Transactions American Institute Mining Engineers*, Vol. XV., p. 25.

The acid rocks, such as diorite, porphyry, and granite, contain large numbers of veins carrying copper ores with quartzose gangue, but they have scarcely ever proved productive in this region. It is an interesting fact that in these acid veins, the surface carbonates and oxides usually change within a few feet into copper glance, and at no very great depth, into the ordinary chalcopyrite, while the limestone veins carry great bodies of oxidized ores to very considerable depths, and change into chalcopyrite without any marked appearance of copper glance.

As Wendt justly remarks, all the important Arizona deposits seem to be true fissure veins, in the sense that they are bodies or masses of ore deposited in the rocks that now contain them, subsequent to the deposition or formation of these rocks.

Great bodies of clay are almost invariably found in conjunction with these veins, resulting, evidently, from the decomposition of the rocks due to the enormous thermal action that has taken place during the deposition of the copper ores. The walls of the Longfellow mine often consist of pure white kaolin, of which Wendt gives the following analysis:

Silica	42.40
Alumina	32.50
Ferric oxide	16.17
Lime	2.10
Magnesia	Trace.
Copper	Trace.
	93.17

The balance of the 100 per cent. was principally moisture.

The four important Arizona copper districts are at present:

The Clifton District.	The Globe District.
The Bisbee District.	The Black Range District.

It is only possible, in this brief sketch, to outline a few of the most important characteristics of these interesting deposits.

The Clifton District, like most of the other copper areas, contains three distinct systems of veins carrying copper.

1. Veins occurring in limestone.
2. Veins occurring in porphyry or felsite.
3. Veins occurring in granite.

The ores of the first system consist mainly of cuprite, in a gangue of compact hematite; and of malachite and azurite, in a gangue of manganese, or wad. An analysis of a characteristic specimen of this cupriferous wad by Professor Mayer yielded:

Cupric oxide	28.39
Manganic oxide	31.24
Silica	24.81
Water	11.87
Ferric oxide and carbonic acid	2.74
Lime	Trace.
	99.05

The most noted mine of this class is the Longfellow. A reference to Fig. 1 will show it to be an almost vertical fissure in stratified limestone, at or near its junction with a strong dyke of felsite.

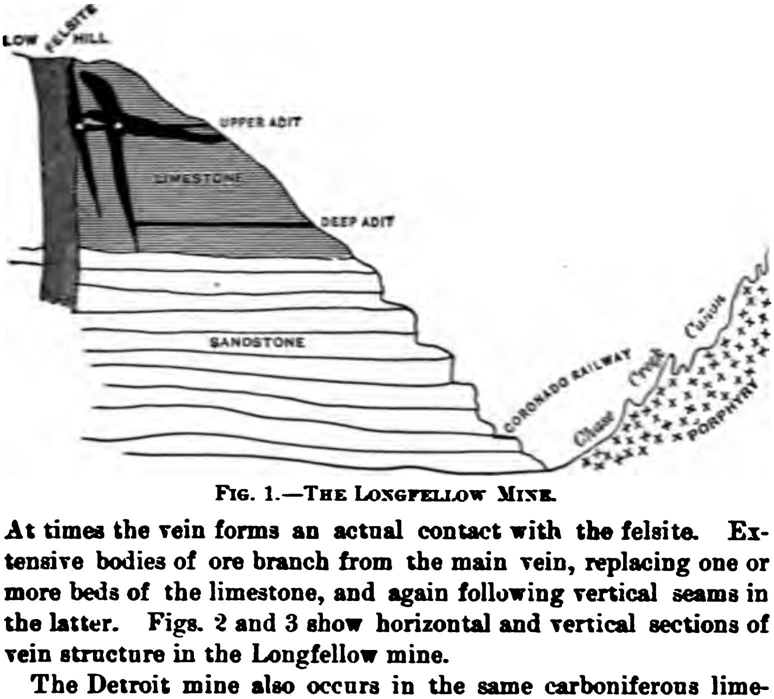


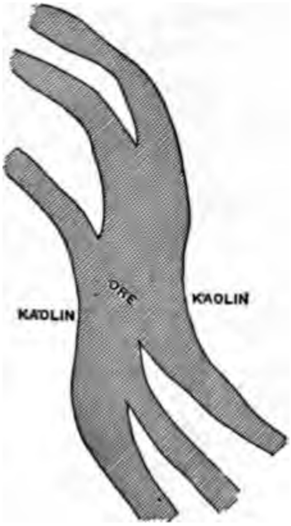
FIG. 1.—THE LONGFELLOW MINE.

At times the vein forms an actual contact with the felsite. Extensive bodies of ore branch from the main vein, replacing one or more beds of the limestone, and again following vertical seams in the latter. Figs. 2 and 3 show horizontal and vertical sections of vein structure in the Longfellow mine.

The Detroit mine also occurs in the same carboniferous limestone, in close proximity to a dyke of fine-grained green eruptive rock. Its ores are mainly azurite and cupriferous wad.

The second class of veins occurs in porphyry, and presents too varied features for detailed description in this connection. One of them is shown on Metcalf Hill, see Fig. 4, where, at the surface, it forms a stockwerk of oxidized veinlets over 100 feet wide in the porphyry, which soon unite into a single vein carrying copper glance, at greater depths deteriorating into unprofitable ores.

Another interesting example of the second system of veins is the Coronado group, in a strong dyke of quartz porphyry, cutting



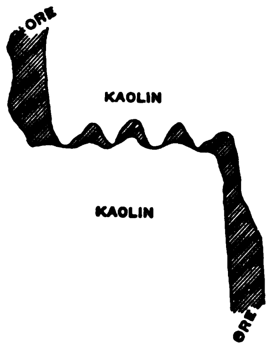


FIG. 8.

FIG. 2.—HORIZONTAL SECTION. FIG. 3.—VERTICAL SECTION.
THE LONGFELLOW MINE.

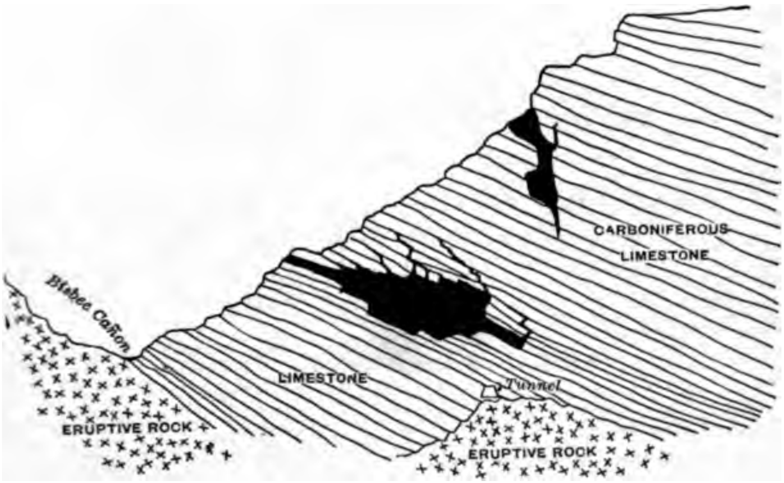


FIG. 6.—BISBEE DEPOSITS.

through syenite and granite, which latter abuts against, and is surrounded by, stratified limestone, as shown in Fig. 5. Near the surface, these veins carry strong bodies of rich copper glance, mainly where the porphyritic walls are strongly decomposed and kaolinized. As depth is gained the rich ore gradually disappears, and at 150 to 200 feet from the surface the vein becomes barren, or contains only sparsely disseminated chalcopyrite.

Two partial analyses, by Henrich, of typical ores of this class show their silicious character:

	I.	II.
Copper	11.17	21.95
Silica	67.00	48.90
Iron	6.91	9.41

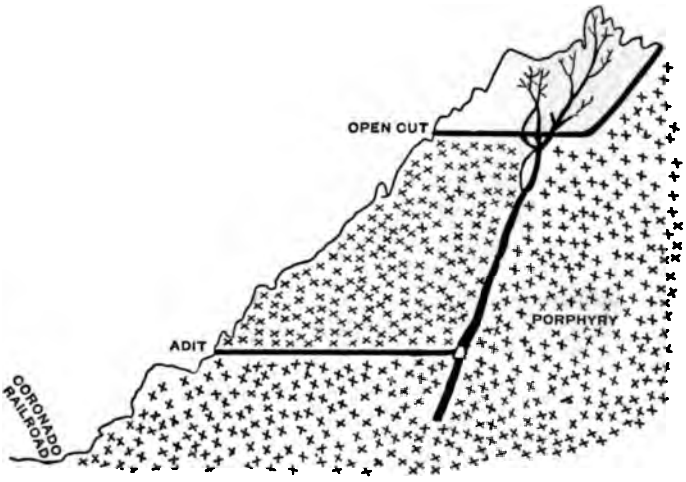


FIG. 4.—SECTION OF METCALF HILL, CLIFTON.

The veins of the third system occur in granite, at a great altitude and in extremely inaccessible situations. They are strong and of good width—5 to 12 feet—and carry copper glance near the surface; but their mineralization is very irregular and they have been little worked.

The Bisbee District is in the Mule Pass Mountains, in Southern Arizona, only 10 miles from the Mexican border.

A great mass of eruptive rock has upheaved the carboniferous limestone, and along the southern contact occurs the Copper Queen group of deposits. (See Fig. 6.) They correspond closely to Von Cotta's "bed-veins." They are not simply "ore-beds," as they send numerous spurs into the walls. These spurs usually follow the planes of bedding of the limestone, and the mode of deposition is still further complicated and obscured by the occurrence of affiliated bodies of ore in the limestone, which were evidently deposited in vugs and caves.

The ore consists mainly of hydrated oxides of iron and alumina, carrying, at present, about 8 per cent. of copper, after undergoing a moderate selection. To a depth of over 400 feet the copper was mainly in the form of carbonates, but as greater depth is gained sulphides are encountered, and a converter plant has just been erected.

The Bisbee black copper, as produced by a single fusion of the oxidized ores with coke, in a water-jacket cupola, is of excellent

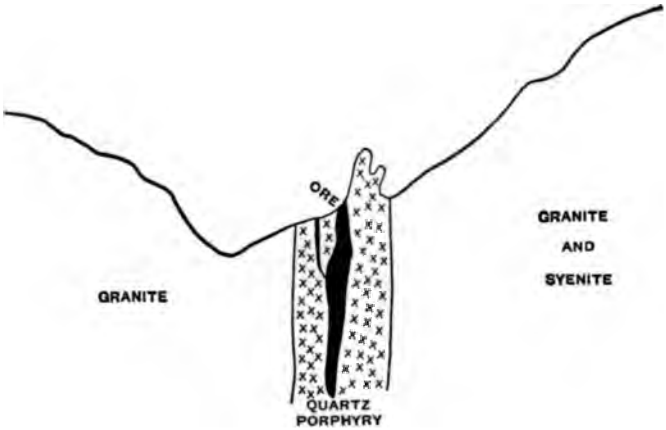


FIG. 5.—VERTICAL CROSS-SECTION OF CORONADO VEIN, CLIFTON.

quality, the following analysis by the Orford Copper Company, representing one lot of 60 tons.

Copper	95.00
Sulphur	0.44
Iron	4.23
Insoluble	0.51
Arsenic	None.
Antimony	None.
	100.18

The Globe District is situated more toward the center of Arizona, on the eastern slope of the Pinal Mountains.

The main ore body that has made this district famous, is situated, as usual, in carboniferous limestone, close to an upheaval of diorite. (See Fig. 7.)

An analysis by Dr. Trippel, of a week's delivery of ore to the furnaces, gives:

Silica	20.23
Ferric oxide	42.10
Alumina	4.15
Loss by ignition	9.75
Oxide of copper	17.12
Magnesia	2.85
Lime	1.12
Oxide of manganese	1.63
	98.95

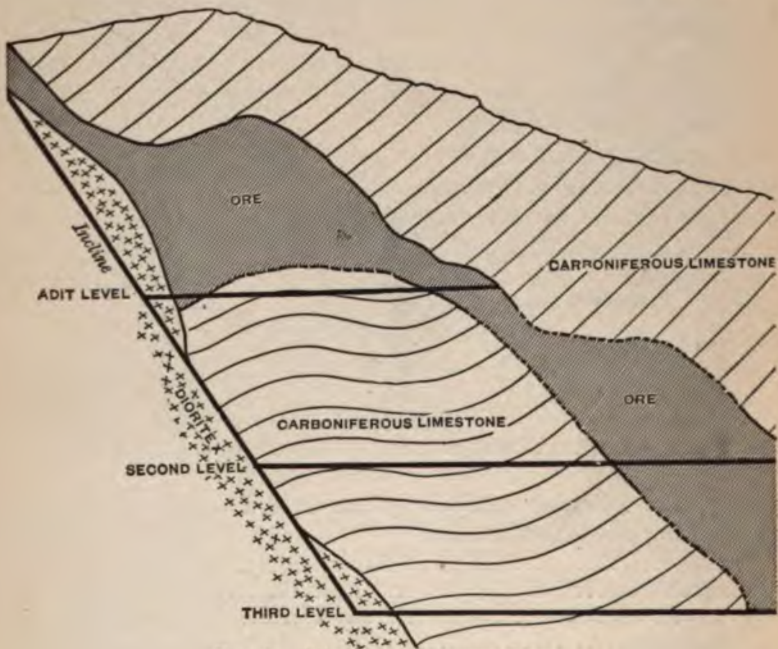


FIG. 7.—SECTION OF GLOBE MINE.

This sample is more ferruginous than the general run of the ore, which usually requires the addition of limestone before smelting.

Very pure black copper is produced by a single fusion with coke, in water-jacket furnaces, the following analysis by Trippel being a sample of two weeks' production, which is, however, slightly above the average in purity:

Copper	99.11
Lead	0.67
Sulphur	0.08
Slag	0.08
Arsenic	Trace.
Iron	Trace.
	99.94

The Black Range Copper District is situated near the center of Arizona, on the eastern slope of the Black Range, and close to the Verde river.

The veins occur near the contact of a belt of diorite and slate, and are of great strength. The non-argentiferous green carbonates and oxides give way to massive pyrite and chalcopyrite at a depth of about 150 feet. These sulphides contain moderate amounts of silver and gold, the oxysulphureted ore (often called black oxide), at the junction of the oxidized and sulphide ores, being often extremely rich in the precious metals. Very extensive and valuable bodies of pyritic ores have been lately developed, and are being smelted and converted on the ground, and a railroad is building to the mine.

The mining districts of Lake Superior, Butte, and Arizona furnish about 95 per cent. of the total copper produced in the United States.

CHAPTER III.

THE SAMPLING AND ASSAYING OF COPPER.

THE first step usually taken in the treatment of an ore of copper is to learn its value by determining the proportion of that metal that it contains. This process is called assaying, as distinguished from chemical analysis, which includes the further investigation as to the general composition of the ore.

We shall confine our discussion in this place to assaying only. The assaying of any given parcel of ore is necessarily preceded by the process of *sampling*, by which we seek to obtain, within the compass of a few ounces, a correct representative of the entire quantity of ore, which may vary in amount from a few pounds to several thousand tons. With rich ores, it will lessen the chance of serious error in large transactions to divide the lot into parcels of not over fifty tons each, and sample each of these lots by itself.

The utmost care and vigilance in sampling and assaying should be required at every smelting works, both in the interest of the works and in that of the ore-seller.

American conditions have encouraged the use of automatic devices for the sampling of ores and mattes, and although there is still a certain prejudice against them in some private works, I believe that they have been adopted by all public sampling works of any standing.

Such works are constantly handling large quantities of rich and very varied material, and it is a matter of absolute necessity to them that their methods of sampling should be above suspicion, and free from the factor of "personal equation" that would be introduced by the employment of a reasoning agent to take the sample.

Automatic samplers, constructed on correct principles, must necessarily attain absolute accuracy, and a sufficiently extended comparison of their results with those obtained by hand-sampling, will satisfy any one of their superiority.

The methods of hand-sampling are too well known to demand description in these pages.

Automatic samplers may be divided into two classes:

1. Those which divert a *portion* of the falling stream of ore, either constantly or intermittently.

2. Those that divert the *entire* ore-stream, for an instant, at regular intervals of time.*

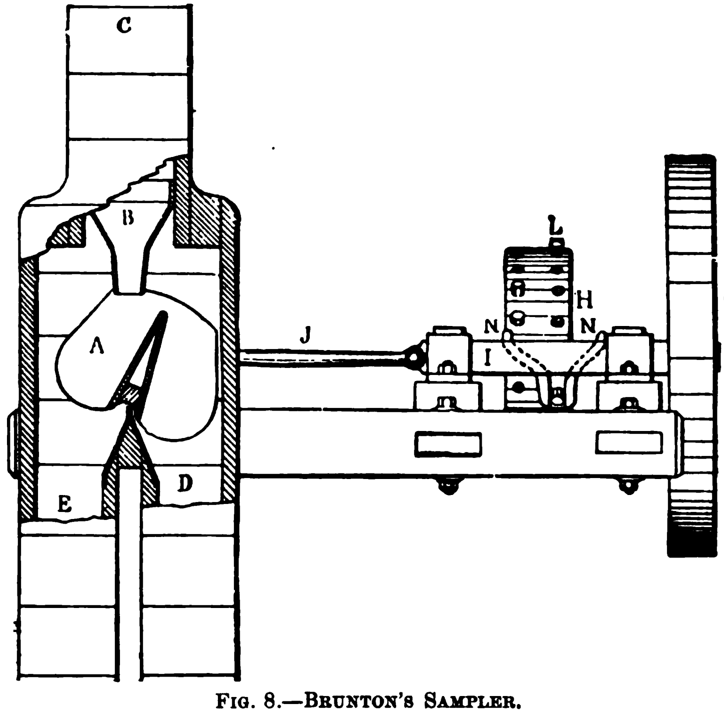


FIG. 8.—BRUNTON'S SAMPLER.

The devices of the first type are very numerous. Some of them are: A cone, or dividing-box, upon which the crusher discharges, and which automatically separates from one-third to one-tenth of the whole. The sample thus obtained can be still further diminished by successive operations on similar, but smaller apparatus, lower down. Or, a wedge is used to separate the falling ore-stream into a very large, and a very small portion.

*The tables, and much of the text that follows, are adapted from Dr. Ledoux's paper on "American Methods of Sampling and Assaying Copper," THE MINERAL INDUSTRY, Vol. I.

Many ingenious machines exist for accomplishing the same end by various means; but none of them have been entirely satisfactory, owing to the tendency of the coarse and fine particles of ore to segregate, and thus to render the ore-stream richer laterally, or in the center. And on different ores the relative position of these rich and poor streaks may vary completely.

Hence, we must turn to the second class of automatic samplers —those that momentarily divert the entire falling ore-stream for a sample. On well-planned machines of this description, foreign substances, such as rags, chips, frozen lumps of ore, etc., produce no effect inimical to accuracy.

BRUNTON'S AUTOMATIC SAMPLER.

This machine deflects the entire ore-stream to the right or left, while falling through a vertical or inclined spout. By a simple

FIG. 9.—BRUNTON'S QUARTERING SHOVEL.

arrangement of movable pegs, in connection with the driving gear, the proportion of the ore-stream thus deflected into the sample-bin may vary from 10 to 50 per cent.; the latter amount only being required in coarse ores of enormous and very variable richness, while for ordinary lump ores, from 10 to 20 per cent. is the maximum required.

Instead of passing the sample-stream of ore into a bin, this system may be still further perfected by leading it directly to a pair of moderately fine rolls, the product of which is elevated to a second similar sampling machine, from which the final sample drops into a locked bin, to be pulverized and quartered by hand.

The two machines are driven at different speeds, to prevent any possible error that might rise from isochronal motion.

A still more recent invention of Mr. Brunton's is the quartering shovel, described in the *Engineering and Mining Journal* of June, 1891.

H. L. Bridgman has invented and introduced an automatic sampler whose principle is so sound and results attained so satisfactory, that I feel obliged to describe it at some length. I make use of a portion of Mr. Bridgman's description and illustrations.*

MACHINE A.

This machine occupies a floor-space of 3 by 4 feet, and has a total height of 7 feet 6 inches. It is self-contained, requiring only to be bolted to the floor and to have feed, discharge, and belt connections made. Fig. 10 shows the machine as it is built, while Figs. 11 and 12 give the diagraphic sections and details, some minor changes and omissions having been made for the sake of clearness. The machine consists essentially of three apportioners, I, II, and III, all driven by the one pulley, X (usually tight and loose pulleys), and three stationary, concentric receptacles, R_1, R_2, and H, so constructed that any material falling into them will pass out through the spouts T_1 and T_2 into the sample-buckets Z_1 and Z_2 or through the spout S, which discharges the rejected portion of the sample. Apportioners I and III revolve in the same direction, apportioner II in the opposite direction; I at about 5, II at about 15, and III at about 45 revolutions a minute. That is to say, each apportioner moves actually three times as fast as the one above it, and in the contrary direction, or, relatively, four times as fast. By the use of this expedient of contrary revolution, the same relative speeds are obtained as though, all revolving in the same direction, the actual speeds were respectively 5, 25, and 125, at which latter speed centrifugal force would become very troublesome.

The upper apportioner, I, consists of two concentric rings, divided by 8 partitions into 8 equal topless and bottomless compartments, L, from each one of which leads an adjustable spout, either as M-1, or as M-2, or as M-D. Set in rotation, spout M-1 would describe a certain circular path, 1-1; spout M-2 a certain other path, 2-2, and spout M-D a third path, W (see Fig. 12).

The intermediate apportioner, II, is merely a conical funnel, having, besides the large outlet W, four vertical shoots, N_1-N_1 and N_2-N_2, through its sloping sides as shown in Fig. 12; each one of these shoots forms one-eighth of the circular paths covered by the spouts M-1 and M-2 respectively.

The lower apportioner III is of the same construction as II and bears the same relation to it that II bears to I.

* *Transactions American Institute Mining Engineers*, Vol. XX., p. 416.

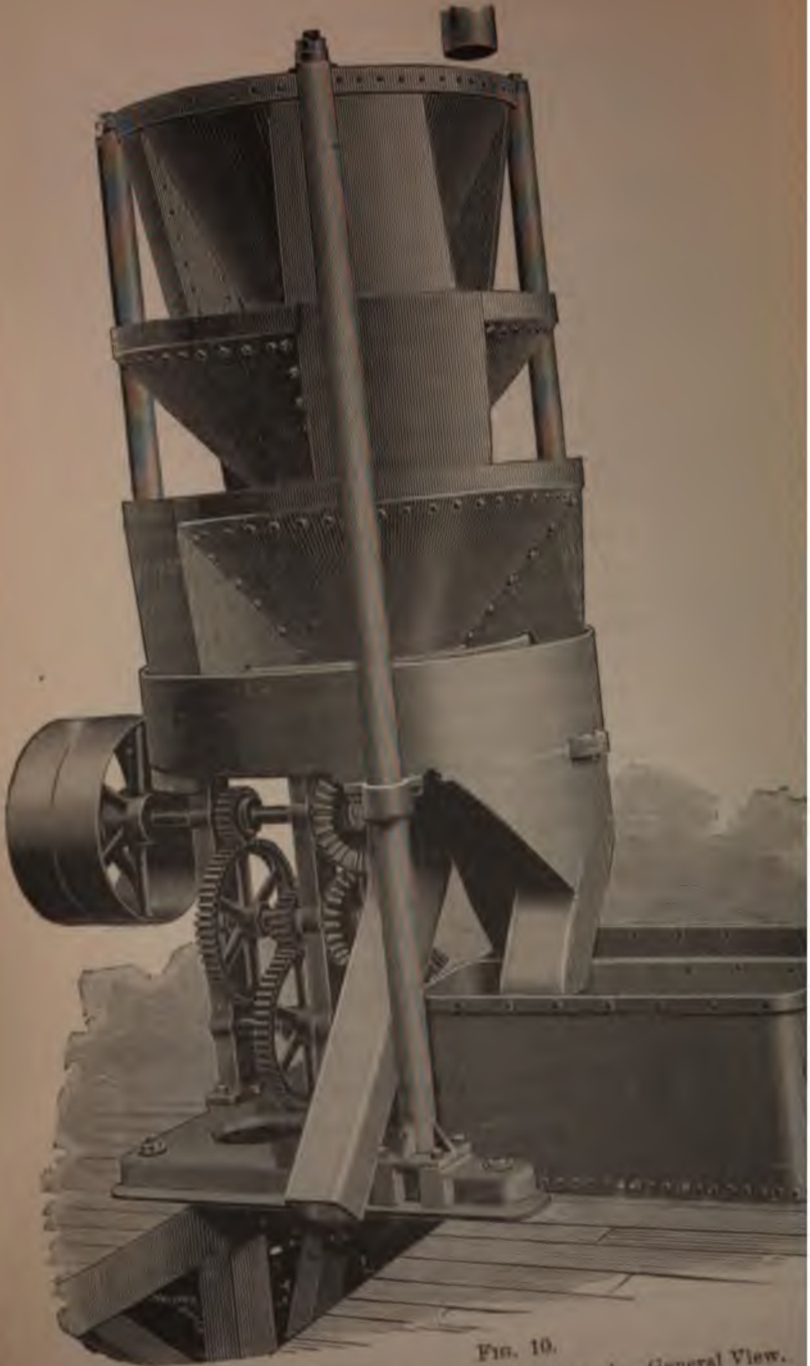


FIG. 10.
Mechanical Ore Sampler. Machine A. General View.

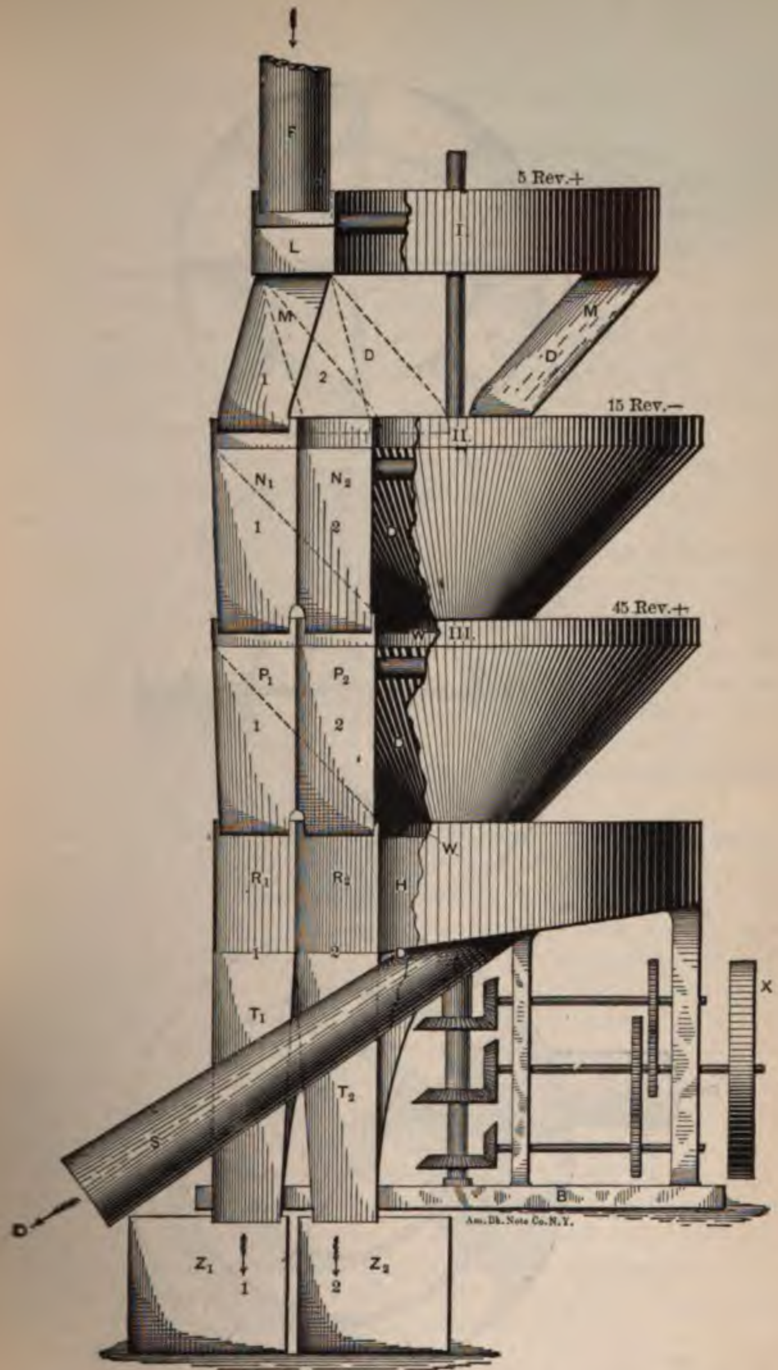


FIG. 11.

Mechanical Ore Sampler. Size A. Total Height, including Sample Buckets, 7 feet 6 inches.

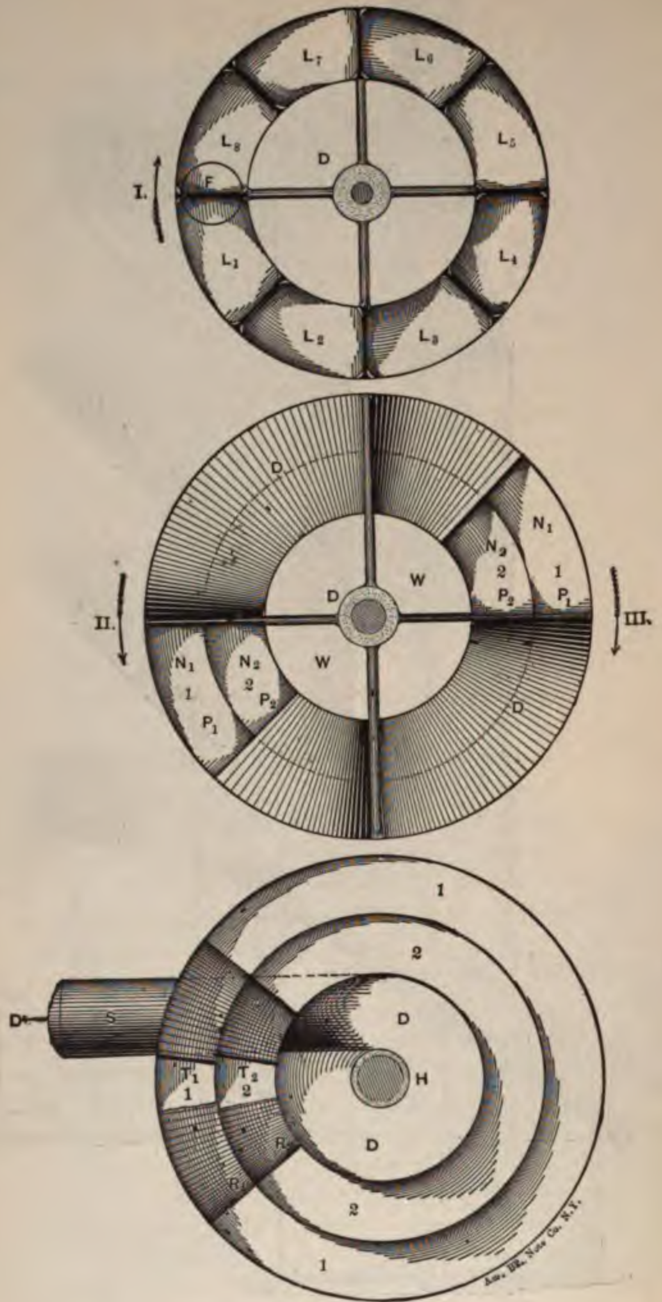


FIG. 12.
Mechanical Ore Sampler. Size A.

An example will best illustrate the operation of the machine. It may be assumed that an original sample of 40,960 pounds (the 960 being added to avoid fractions) is to be put through the machine; that the time required will be one hour; that the speed of the machine is such that the upper apportioner, I, will make 320 revolutions in that time, and finally that the ore is of such grade and character as to only require the smallest sample that the machine will give. Under these conditions, one of the spouts, M_1, would be set as M-1, one (the opposite one) as M-2, and the remaining six as M-D (Fig. 11).

The flow of material, previously crushed to below one inch in size, would then be started through the feed-spout, F, and the machine set in motion.

It is evident that at each revolution one 320th part of the whole lot, or 128 pounds, will pass through the feed-spout F. Of this amount six-eighths, or 96 pounds, will be discarded by the six spouts, M-D, passing down through W, W, H, and so through the spout S and out of the machine, while one-eighth of the 128 pounds, or 16 pounds, forming the first cut of the first or outer sample, will pass through the spout M-1, and the remaining one-eighth, or 16 pounds, forming the first cut of the second or inner sample, through the spout, M-2.

These two first cuts will proceed side by side, by separate paths, through the same series of operations, and whatever applies to the one, applies equally to the other; it will, therefore, suffice to follow the first sample. This one-eighth, or 16 pounds, having been cut from the mass by the partitions of the compartment L_1, of which M-1 forms an extension, will drop nearly vertically through M-1 on its way to the sample box, Z_1. As it leaves the spout, M-1, during the one-eighth of a revolution that is occupied by the said M-1 in passing beneath the feed-spout, F, it will be intercepted by the intermediate apportioner, II, which in the same time will have made one half-revolution (relatively to I).

Since the vertical shoot, N-1, occupies one-fourth of the semicircumference of II passing beneath the spout, M-1, it follows that one-quarter of the 16 pounds, or 4 pounds, will drop vertically through this shoot as the second cut of the first sample. The remaining three-quarters, or 12 pounds, will pass down the sloping sides of II and be discarded through W, W, H, and S.

In precisely the same way, the second cut of 4 pounds will be quartered by the lower apportioner, III, 3 pounds being discarded and 1 pound, as the third and final cut, passing through the vertical shoot P_1 and the spout T_1 into the sample bucket Z_1.

In the same way a 1-pound portion, as the third cut of the second sample, will find its way to the bucket Z_2.

This series of operations will occur at each revolution of the upper apportioner; and at the end of the hour each of the buckets Z_1 and Z_2 will contain 320 portions of 1 pound each, or a total final sample of 320 pounds, these two total samples being as independent of each other as though made at different times and places. It will of course rarely happen that this theoretical exactness of weights will obtain, which point will be considered later. Should the ore be of higher grade or more irregular in character, two or three or four of the spouts, M, may be set for each sample, giving final samples of 640, 960 and 1,280 pounds respectively.

It will be noticed that only the discarded part of the sample is touched by the machine, the retained portions dropping nearly vertically and practically freely through the machine, until, *in a finished condition*, they reach the stationary receptacles R_1 and R_2, or the sample-buckets Z_1 and Z_2. The machine can have had, therefore, no influence on the constitution of the samples, and "coarse" and "fine" must be contained therein in the same proportion as delivered by the feed-spout F.

It may be remarked in passing that the finer the material the slower the feed, and the greater the speed of the machine the greater will be the distribution and, presumably, the better the samples. The conditions above given, however, are easily attained, depending only on the crushing capacity at disposal, and have been found by experience to give satisfactory results, it being particularly desirable not to use a much higher speed. For light or wet ores it may be neceessary, in order to avoid an accumulation of material in the machine, to reduce the speed to half that given. This lower speed may of course be used for heavy materials also, the only practical difference between the higher and lower speeds (aside from the influence of centrifugal force) being the difference in the number of cuts made by the machine.

In lump ores, it is difficult to obtain a correct sample, even for moisture, without some preliminary crushing, and to save labor it is best to use a portion of the large sample from the automatic sampler for this purpose; the accurate weighing of the entire ore parcel being postponed until just before, or after, the sampling, and the portion reserved for the moisture determination being placed in an open tin vessel, contained in a covered metal case, having an inch or two of water on its bottom, in which the sample tins stand.

From one-fourth to one-half pound of the sample is usually weighed out for this determination, and dried under frequent stirring, and at a temperature not exceeding 212 degrees. While it is always important to keep within the limit of temperature just mentioned, it is especially the case with certain substances which oxidize easily. Among these are finely divided sulphides, and above all, the pulverulent copper cements obtained from precipitating copper with metallic iron from a sulphate solution.

Such a sample, containing actually 5½ per cent. of moisture, showed an increase of weight of some 2 per cent. on being exposed for thirty minutes to a temperature of about 235 degrees Fahr.

Certain samples of ore—especially from the roasting furnace—are quite hygroscopic, and attract water rapidly after drying.

In such cases, the precautions used in analytical work must be employed, and the covered sample weighed rapidly, in an atmosphere kept dry by the use of strong sulphuric acid..

The sampling of the malleable products of smelting, such as blister copper, metallic bottoms, ingots, etc., can only be satisfactorily effected by boring a hole through a certain proportional number of the pieces to be sampled.

Where such work is only exceptional, an ordinary ratchet hand-drill will answer, but in most cases, a half-inch drill run by machinery is employed.

The chips and drillings are still further subdivided by scissors, and as even then it is difficult to obtain an absolutely perfect mixture, it is best to weigh out and dissolve a much larger amount than is usually taken for assay, taking a certain proportion of the thoroughly mixed solution for the final determination.

Many of the smelters are too careless in the sampling of their metallic products. At the public sampling works in New York, where much copper in metallic form has been shipped abroad for refining and separation, the following precautions have been found necessary to ensure uniform results.

With copper bars that are tolerably uniform, and free from precious metals, every fifth bar is bored halfway through, on opposite sides.

In sampling argentiferous bars, every bar is usually bored twice. If the bars carry appreciable quantities of gold (and always in the case of anodes) the borings are melted and granulated, or recast into a sample bar, which is again bored.

In sampling and assaying matte for shipment, it must be remembered that in the long journey from the West there is always a certain loss in weight. This is especially the case when the matte is shipped in pigs (in bulk), and there are one or more transfers. The pigs grind against each other, producing a considerable amount of powder, while the brittle edges and corners are badly chipped. In the hurry of transfer it is almost impossible to have the cars that are emptied cleanly swept, and a loss always occurs, which, in former years, I have been inclined to put at 1 per cent.*

Under somewhat similar conditions, Ledoux found a loss of 0.8 per cent. on a lot of about 500 tons of matte, shipped from the West in bulk and transferred once.

Matte crushed and sacked may undergo a slight shrinkage from sifting, or a still more serious one from torn sacks, if hooks are used in handling it. Ledoux gives the following table as a good average result where care in sampling and sacking is used at the smelter and the material is crushed and sacked. It represents various monthly shipments of matte from a Western smelter to a public sampler in New York, and shows the weights and assays at each end of the line.

Mine Weight. Pound.	Final Weight. Pound.	Difference. Per cent.	Mine Assay. Per cent.	Final Assay. Per cent.
774,277	773,256	0.13	54.92	54.83
805,821	804,552	0.15	57.49	58.03
402,779	402,644	0.03	55.97	55.33
403,458	404,146	0.17	55.97	56.20
420,886	420,178	0.16	51.50	51.80
402,302	401,706	0.14	55.66	56.05
403,604	403,963	0.08	55.21	54.26
421,260	420,892	0.08	55.04	55.47
Average..	....	0.07	55.22	55.25

There is a difference of 0.07 per cent. against the mine in the weights, and of 0.03 per cent. in favor of the mine in the assays.

* In one instance a carload of matte weighed 40,000 pounds. The matte contained 60 per cent. copper, worth, at that time, 10 cents per pound, and no precious metals worth separating. A loss of 1 per cent., therefore, means a money loss of $24 per carload, which, under the conditions referred to, would not pay the cost of sacking.

Some of the matte given in the last table contained silver. The following statement shows the results of the determinations of this metal.

Mine Assay. Ounces.	Final Assay. Ounces.	Difference. Ounces.	Average Difference.
53.20	52.90	0.30	
53.60	53.26	0.34	(Against the Mine)
56.48	57.08	0.64	0.85 Ounces
22.29	22.92	0.63	Per Ton.
54.53	52.20	2.33	

On 88 carloads of matte and bars shipped by the Pennsylvania Salt Manufacturing Company to a New York sampling works, the average total discrepancy was 0.03 per cent. copper, and 0.08 ounces silver per ton of 2,000 pounds.

It has been a matter of great importance to smelters and miners in this country to learn the exact system of weighing and sampling practiced in England, in order that they might obtain some clue to the heavy discounts they are often obliged to bear, both in weights and assays. The exportation of copper mattes or other similar products from the United States to England, for refining, has pretty much ceased, as they can be treated more profitably at home; but as there are other countries which will doubtless continue shipping copper-bearing material to Swansea for a long time to come, it may be useful to describe the difference between the English method of weighing and assaying, and our own. I again quote from Dr. Ledoux:

"In the United States, the public samplers—at least those in the East—employ "sworn weighers," who have gone before a notary public and taken an oath of office. Ore in bulk is weighed on platform scales in barrows, or small wagons, holding 500 to 1,500 pounds. The weight is taken on a rising beam, which amounts to an allowance of one-eighth to one-fourth pound per load. Ore or matte in bags is always, where practicable, weighed on beam scales, six to ten bags being taken for a draft. No returns are ever based on carload weights on track scales.

"The settlement is always based on the acutal weight ascertained as above, with no allowance for draftage, etc., and upon the actual percentage of copper found by analysis, less the arbitrary deduction of 1.3 per cent. of fine copper. This arbitrary deduction is the result of agreement between the copper smelters and producers of the United States, and is supposed to represent the average loss in smelting. In refining bars, there is, of course, no such loss as 1.3 units, and the smelter is the gainer; while in leady mattes or base ores there may be a considerably greater loss than 1.3 per cent. In America, the smelter protects himself in the price he bids or in the refining toll he charges, instead of asking the assayer to find out for him, in each case, what his loss is likely to be—which is what the Cornish assay attempts to do.

"I am indebted to the Liverpool Wharf Company for the following table, representing its experience with some 2,500 tons of matte from America:

ORDINARY COPPER MATTE.

Shipping Weight. Tons.	Landing Weight. Tons.	Loss. Per cent.
1,355[illegible].	1,354[illegible].	0.2

ARGENTIFEROUS COPPER MATTE.

Shipping Weight. Tons.	Landing Weight. Tons.	Loss. Per cent.
1,139[illegible].	1,130[illegible].	0.75

"Difference between American assay, after deduction of 1.3 per cent. and Cornish assay, 1.79 per cent. copper, more or less.

"Difference between American and English assay for silver, 0.30 ounces per ton of 2,240 pounds.*

* Occasional shippers are sometimes embarrassed by the unfamiliar weights and money used in the English shipping returns.

The English employ the long ton of 2,240 pounds, which they divide into 20 hundredweight, each hundredweight containing 4 quarters of 28 pounds each.

When these weights are to be multiplied by a specified number of pounds, shillings, pence,and farthings, it forms an exceedingly fascinating problem for an idle day; the very uncertainty of the result adding, in no small degree, to the interest of the operation.

I append a recent example of a shipment of some high-grade material to London, giving the final results in both English and American weights and values.

The English returns gave 29 tons, 17 hundredweight, 3 quarters, 27 pounds, at £217 4s. 3½d. per ton.

The translation into American gives 33,487 tons (of 2,000 pounds) at $1,051.32 equals $35,205.55. (Thirty-three and four-hundred eighty-seven-thousandths tons at ten hundred fifty-one dollars and thirty-two cents, makes thirty-five thousand two hundred and five dollars and fifty-five cents.)

Those interested in the higher mathematics may enjoy calculating the English returns.

"In my opinion, an average deduction of fourpence per unit of copper made by British buyers purchasing in the United States, free on board in New York, and selling again at English terms, will be sufficient to cover the difference caused by loss in transitu and by the employment of the Cornish essay.

"Considerable of the loss in weight can be avoided if matte is shipped in barrels instead of bags. Good kerosene barrels can be purchased at about 85 cents apiece. Glucose barrels are too sticky."

*THE ASSAYING OF COPPER.

American assayers and chemists† are accustomed to exercise entire freedom in their selection of method employed for the determination of the constituents of any material submitted to them. It is only required of them that their results be correct. Consequently, they do not make use of the Cornish fire-assay, which is not properly a method for the determination of the exact amount of copper contained in an ore, but rather an ingenious adaptation of metallurgical processes to laboratory conditions, and which is intended to show the amount of copper the smelter may expect to produce from the ore in question.

On the whole, it is decidedly favorable to the smelter; as on any ordinary sulphide material of tolerable richness, it rarely gives so high a result as the analytical assay, less our 1.3 per cent. arbitrary deduction. And as 1.3 units has been found, by long experience, to be a sufficient deduction to cover the actual metallurgical loss in ordinary furnace material, the inference is obvious. Moreover, it introduces an unfortunate element of uncertainty into all transactions between miner and smelter, as the different chemists seldom agree exactly in this assay, and frequently differ widely. With us, a variation of 0.2 per cent. is sufficient to call for adjustment.

In assaying slags for copper it should be borne in mind that even after apparent complete decomposition by acid, the insoluble residue

* Not feeling myself competent to treat exhaustively of the improvements made in copper-assaying during the past fifteen years, I have availed myself of the kind assistance of several well-known chemists. Their names will appear in connection with the sections that they have prepared for me.

† In England all analytical chemists are styled assayers. In the United States the term assayer is applied to those chemists who busy themselves chiefly with the determination of the valuable metals.

may still contain an appreciable amount of copper. Hence, acid slags should frequently be treated by fusion with alkaline carbonates, as is customary in analyzing silicates.

The assayer of the present day will find it convenient to be thoroughly familiar with the three methods that are sufficiently accurate and concise to be practically employed for the quantitative determination of copper in all classes of material. These are:

1. The electrolytic assay.
2. The *improved* cyanide assay.
3. The iodide assay.

To which may be added, under exceptional conditions,

4. The colorimetric assay.
5. The Lake Superior fire-assay.

I am aware that the statement that the first and third of these methods are practically equal, in scope and exactness, will be received with incredulity by many experienced chemists. It took me some years to learn that the *improved* cyanide method gave results almost equal to the battery assay, when executed with equal skill; and it is only on a recent visit to England that I began to appreciate the extent to which the iodide assay has replaced the electrolytic method in that conservative land.

In some of the largest and most carefully conducted works in England, and especially in one smelter, that has a very extensive electrolytic refining plant of its own, the iodide assay is employed to the exclusion of all other methods, and, as I was assured by the chemist in charge of the laboratory, with more satisfactory results than the battery assay.

At least two among the best public laboratories in this country are now making all their copper determinations by this method, and I have letters from the principals of each of these offices, stating that they intend using it in place of the battery assay.

The battery jars are always a nuisance; and even where a constant current can be obtained from the electric-light wires, the iodide assay seems to be preferred by several chemists who have a large amount of work to do, and who have given it a fair trial.

Hence, I feel that it will be useful to the profession to give the details of the operation at length, both as practised in England, and as modified by one of our most experienced American assayers.

I.—THE ELECTROLYTIC METHOD, OR BATTERY ASSAY.

This is suited to nearly every class of material and every percentage of copper, from the highest to the lowest, and, owing to its ease of execution and extreme accuracy, has already largely supplanted the ordinary analytic methods, and bids fair to do so altogether in all important cases. Among those assayers who do not yet practise it, there seems to be an impression that it is difficult of execution, and in several cases under the author's observation it has been abandoned after a few futile efforts. In these instances there must have been some direct violation of the laws governing the generation and transmission of electricity—it being always the battery that was complained of—and as a similar though usually a much more extensive and complicated form of battery is under the charge of every telegraph operator, the disappointed assayer should feel encouraged to persist.

Messrs. Torrey & Eaton have also investigated the effect of various substances upon the battery assay, and have arrived at the following results, which are not quite so favorable as the author's experience in practice has been:*

"*Silver*, when present in any considerable proportion—from 1 to 3 per cent.—gives too high a result. It should always be removed by hydrochloric acid.

"*Bismuth*, even when present in small quantity—½ per cent.—is partly or wholly precipitated with the copper, and must consequently be determined analytically in the deposit. A solution of .970 gram copper, .030 gram bismuth, gave 97.9 per cent. copper instead of 97 per cent.

"*Lead*, derived from the resolution of sulphate of lead (if present) by the wash-water, is partially precipitated with the copper. This applies only to large percentages of lead.

"*Zinc and Nickel* do not interfere in quantities up to 30 per cent.

"*Arsenic* precipitates partly *with the copper*, and not *after it*, as has been supposed. It gives a bright deposit, but may be found in considerable quantity in the precipitate, before the solution is free from copper. After complete precipitation of the copper, therefore, the deposit should be titrated with cyanide of potassium."

The following description has been written for me by Mr. Francis

* Mr. Sperry's experience shows that with proper precautions these unfavorable results may be completely avoided

L. Sperry, analytical chemist, and for five years chemist to the Canadian Copper Company, at Sudbury, Ontario.

The scheme, as given, is intended to present the details in as practical and concise a manner as possible without going beyond the province of this work. Those who desire to study more closely the electrolysis of other metals, and also the treatment of copper in oxalate solutions which can advantageously be made use of, are referred to the admirable work of Dr. Classen on "Quantitative Chemical Analysis by Electrolysis," and also "Electro-Chemical Analysis," by Prof. Edgar F. Smith.

THE DETERMINATION OF COPPER BY ELECTROLYSIS.

Of the various methods the chemist has in hand for the determination of copper, the electrolytic method presents some advantages over other recognized forms. It permits of reliable, clean, and rapid work, and enables the chemist to remove copper from a solution completely, in the presence of other metals, which may subsequently be determined in the same solution.

The requirements are clean platinum cathodes and anodes and a uniform current of electricity of known strength.

Take, for a weighed sample, one-half a gram copper matte, one or two grams copper ore, depending on the richness of the ore, and two or three grams for slag.

In preparing the samples they should be passed through an 80 mesh sieve. Weigh out on an accurate chemical balance.

After weighing the samples in duplicate on watch glasses, transfer carefully to No. 2 beakers, slightly moisten with cold distilled water, add 25 c.c. strong nitric acid and 10 to 15 drops of strong sulphuric acid. The beakers should be covered with watch glasses and set on the sand bath, where they are heated until the nitrous acid fumes have all passed off and the sample is in solution. Wash the watch glasses down into the beaker, and evaporate the solution to dryness.

When choking white fumes appear, set the beakers one side to cool. The copper is now in the form of sulphate. Moisten the mass in the beakers with dilute nitric acid (1.20 sp. gr.), using about 6 or 7 c.c.; add 4 drops of sulphuric acid, 40 c.c. of water, and heat on sand bath until the mass is in solution. Filter off the insoluble matter (which should be examined to see if there may be copper left in the residue undissolved), reserving the filtrate in a No. 1 beaker. The solution is now ready to be electrolyzed.

The electrical energy necessary to electrolyze a copper solution is furnished by various batteries of reliable manufacture. If there is an electric light plant at hand, the wire, properly insulated, can be run through the laboratory and by means of a resistance coil the current can be reduced in strength sufficiently to permit of quantitative electrolytic determinations. The Grenet, Gravity, or Grove cell batteries will be found well adapted for generating the necessary strength of current also. The Grenet cell loses its intensity after long use. The Gravity cell is very likely to act un-

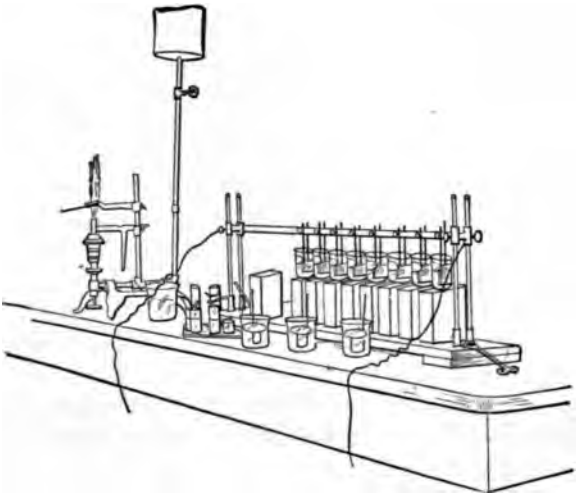


FIG. 13.—RACK FOR BATTERY ASSAY.

satisfactorily on account of local action setting in, causing polarization of the electrodes, and the electrical energy ceasing entirely. The Grove cell requires more care than either of the others spoken of, but the electromotive force is certain to act for as long a time as is necessary for the deposition of the metal, as the copper solutions are set on the battery at night and removed on the following morning, usually.

It is best to have a surplus of electrical energy for the electrolysis, although too strong a current must be guarded against.

Three Grove cells, freshly made up, will furnish current sufficient to electrolyze six to eight copper solutions, none of which contain more than .5 gram copper in 1 gram of sample.

Four cells will electrolyze eight to ten solutions, and five cells, ten to twelve solutions.

A convenient arrangement for supporting the cathodes and anodes for as many as twelve simultaneous determinations of copper is shown by the illustration (Fig. 13).

The rods are $\frac{5}{8}$ inch square by 39 inches long. Holes for the insertion of anode and cathode rods are $3\frac{1}{2}$ inches apart and $\frac{1}{16}$ inch in size, while through the side of the brass rods a milled screw sets against a flexible brass shoe, which binds the cathode and anode platinum rods securely in position. The brass rods, $\frac{1}{2}$ inch apart, are supported on glass pillars, and can be raised or lowered as required.

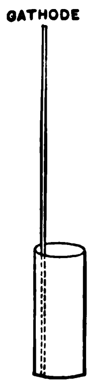


FIG. 14.

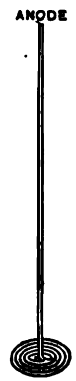


FIG. 15.

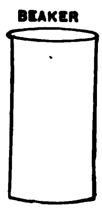


FIG. 16.

The most convenient form of cathode is a plain platinum cylinder $2\frac{1}{2}$ inches long, 1 inch diameter, and the rod that supports it is $4\frac{1}{2}$ inches long. It weighs about 16 to 18 grams (Fig. 14).

These cathodes may seem large, but for general class of work on high and low grade copper ores and mattes they will be found a very convenient size, as they offer a large surface for the deposition of copper, whereas, if they were smaller, the copper would frequently be deposited in spongy form, and there would be loss in weighing.

The anodes are platinum wire of size to fit $\frac{1}{16}$-inch hole, made in the form of a concentric circle, from the center of which the rod stands out 7 inches (Fig. 15).

The diameter of the coil is 1 inch. By this arrangement of the anode there is a uniform evolution of gas throughout the solution, and the inside as well as the outside of the cathode is evenly electroplated with copper.

The cathode should not be completely immersed in the solution to be electrolyzed. When it is supposed that all the copper is deposited, immerse the cathode deeper in the solution and let the current run one-half hour longer. Any deposition on the clean surface will show at once that copper remains still in the solution. If the copper is all deposited, loosen the anode and carefully remove it and the beaker. Wash the cathode quickly into a clean No. 3 beaker with distilled water, immerse in pure alcohol, and gently ignite in flame until dry. The copper should be a pink rose color. Weigh as soon as cooled to the temperature of the room.

The current should not be passed through the solution longer than is necessary to effect the complete deposition of the copper, as secondary reactions are liable to set in.

When there has been a separation of copper in a nitric acid solution alone, the solution should be siphoned off into a clean beaker without interrupting the current, and the cathode washed with pure water, otherwise the nitric acid will dissolve some of the deposited copper into the solution again. Too much nitric acid will keep the copper in solution. Too much sulphuric acid will cause the copper to deposit in spongy form.

Too strong a current will cause loss by too great evolution of oxy-hydrogen gas, the copper will deposit dark colored, and if zinc is present,it will deposit on the copper.

By using deep beakers (Fig. 16), there will be scarcely a perceptible loss of solution by the evolution of gas, as the sides of the beaker should be carefully washed down half an hour before removing the cathode to weigh.

The secondary reactions to be guarded against in passing the current longer than is necessary to deposit the copper, and also in having the solution of proper strength of acid, are the conversion of the nitric acid into ammonia and the formation of ammonia sulphate, so that if the deposition of copper were done in the presence of iron and zinc, these metals would be deposited on the cathode as hydrated oxides.

With the conditions described above conformed to, copper is completely deposited and removed from solutions containing iron, alumina, manganese, zinc, nickel, cobalt, chromium, cadmium, lime, barium, strontium,and magnesium.

In the laboratory of the writer it has been customary to make electrolytic separations of copper and nickel daily for the past five years, and in every case with unvarying accuracy. The copper was removed completely in the presence of nickel, iron and zinc, and these elements subsequently determined in the same solution.

Examples could be given *ad infinitum*, but a few will be given of the most characteristic.

		Sample taken. 1	Sample taken. 2
SLAG	1 gm.		
	Copper,	0.42%	0.42%
	Nickel,	0.41%	0.40%
COPPER ORE	1 gm.		
	Copper,	8.44%	8.43%
	Nickel,	3.33%	3.35%
NICKEL ORE	1 gm.		
	Copper,	1.23%	1.24%
	Nickel,	8.62%	8.63%
COPPER MATTE	.5 gm.		
	Copper,	33.45%	33.46%
	Nickel,	15.75%	15.73%
NICKEL MATTE	.5 gm.		
	Copper,	16.76%	16.78%
	Nickel,	21.23%	21.25%

In each case the nickel was determined electrolytically in the same solution from which the copper was removed.

By carefully noting what are the best conditions, as there is a certain limit within which variation in the treatment of miscellaneous samples is warranted, most any novice will find electrolysis a simple and accurate method for the estimation of copper.

Having noticed a novel and most cheap and convenient apparatus for electrolytic assaying in the laboratory of Messrs. Von Schulz & Low, in Denver, Mr. Low has been kind enough to furnish me with the following description of the same, and of his method of using it.

IMPROVED APPARATUS FOR THE DETERMINATION OF COPPER BY ELECTROLYSIS.

This apparatus was devised as the result of experiments undertaken with a view of accelerating the electro-deposition of copper in analytical work. It possesses the merit of simplicity and rapidity of action, requiring much less care than a battery, and depositing the copper in good, reguline condition in about one-third of the time.

It consists of a small crystallization dish, or beaker, about two inches in diameter, in which a stout glass tube, A, is held by the support B (Fig. 17). Two short glass tubes, passing through suitably shaped pieces of cork, are drawn together with rubber bands so as to hold the large tube firmly, and yet permit of its being easily raised or lowered as required. The lower end of the tube A is ground squarely across and covered with a parchment-paper diaphragm, which is attached as follows: A piece of stout parchment paper, about two inches square, is thoroughly wetted, and the superfluous moisture wiped off. If the paper is thin, two thicknesses may be used. The paper is now pressed over the end of the tube, and secured tightly in place with a rubber band wound around as near the end of the tube as practicable. The loose edges of the paper are cut away close to the rubber with a sharp knife, and the joint is made water-tight by means of a little melted paraffine, applied with a brush. It requires but a few minutes to attach a diaphragm that will serve for several determinations. The tube A is provided at the top with a stopper, through which passes an amalgamated zinc rod C, reaching nearly to the bottom. A small groove is made in the side of the stopper to permit the escape of gas.

In the bottom of the outer cup rests a platinum electrode consisting of a circular base, about one and five-eighths inches in diameter, supporting a series of four concentric walls, about one-half inch high. D is a stout platinum wire extending up out of the cup. There is attached to its lower end where it joins the body of the electrode, a piece of platinum foil reaching up a short distance, to increase the depository surface, and prevent a powdery deposit on the wire. The entire electrode is made of thin platinum foil, soldered with gold. It weighs about eight grams, and exposes (including both sides), about twenty square inches of surface. The zinc rod passing through the stopper must be amalga-

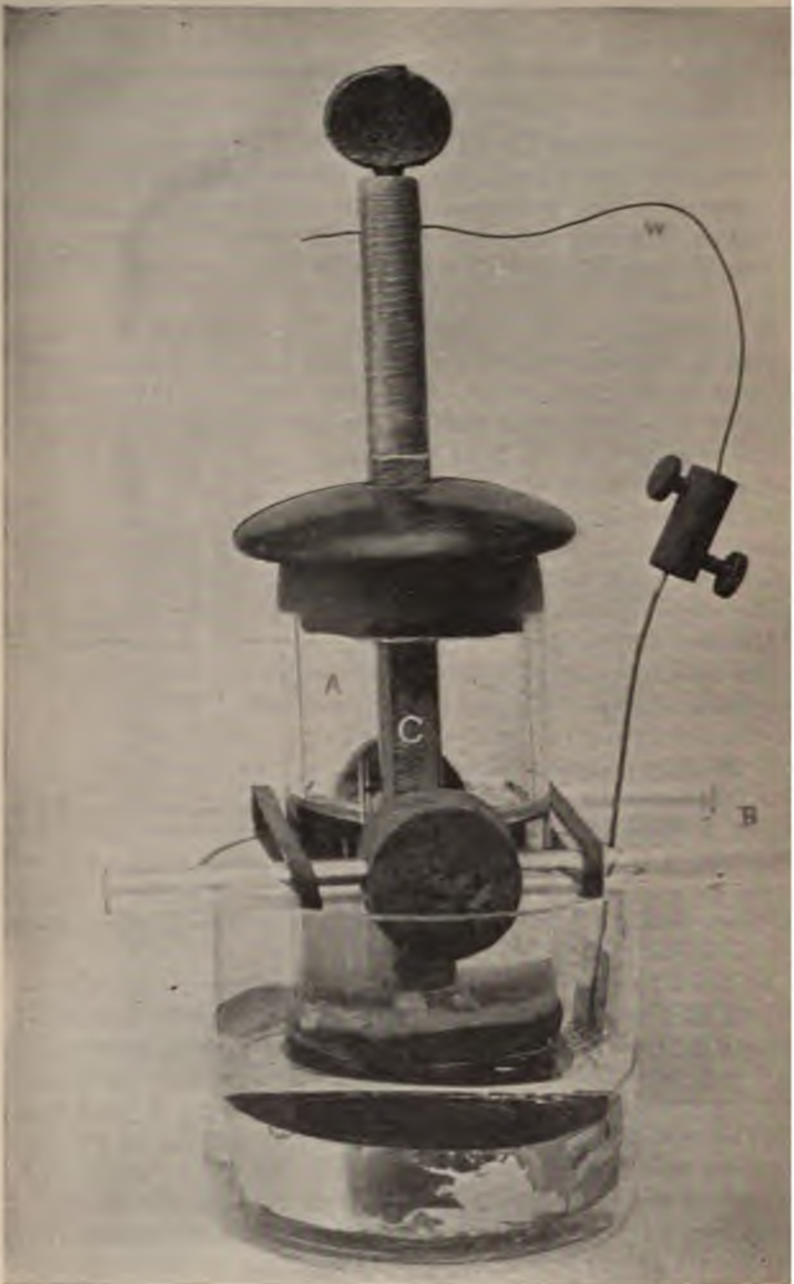


FIG. 17.—LOW'S ELECTROLYTIC APPARATUS.

mated with mercury, and a short copper wire is provided to connect the zinc with the wire D of the platinum electrode.

USE OF THE APPARATUS.

The solution for electrolysis should not contain more than one gram of copper, which may be present either as sulphate or nitrate. The free acid should be neutralized with ammonia, and then the solution made acid with a slight excess, say one c.c., of strong nitric acid. The apparatus being taken apart, the copper solution is placed in the outer cup, which it should be made to fill to the depth of about three-quarters of an inch. The platinum electrode, having been ignited, cooled, and weighed in the usual manner, is now put in place. The tube A is now about half filled with a cold mixture of one part strong sulphuric acid and four parts water, and the stopper and zinc rod inserted. Finally, the tube is placed in the holder and adjusted in the cup so that the diaphragm just rests upon the surface of the copper solution. The latter should be cold, or nearly so. Upon connecting the zinc with the wire D, the deposition of the copper begins at once, and is finished in from one to two hours, according to the amount of copper present. The apparatus requires but little watching. It is well to raise and lower the platinum electrode slightly, to keep the solution well mixed.

After the solution has become perfectly colorless and the operation appears to be finished, the platinum electrode is removed, dried, and weighed, as follows: Without disconnecting them, the tube B and the electrode are lifted together from the glass cup, and while the tube is returned to its place, the electrode is immersed in a beaker of water alongside. If considered desirable, the electrode is rinsed with a little water as it is taken from the cup, but the copper solution is usually so exhausted that the few drops left adhering to the electrode are of no importance. The electrical connection is now broken by detaching the wire W, and the electrode is further washed, first with water and then with alcohol, and finally dried in the usual way. It is now ready for weighing, to facilitate which, a leaden counterpoise may be made, weighing a trifle less than the electrode, so that the difference may be quickly noted by the rider of the balance; and accordingly, in weighing the deposited copper, it is simply necessary to deduct the amount previously determined by the rider. The electrode, after weighing, is cleaned with nitric acid, and again ignited and weighed, and replaced in the battery for a short time, to see if any traces of copper remain in the solution.

Arsenic and antimony, if present in the copper solution, exercise, of course, their usual interference. In the treatment of an ore, the copper is best obtained in the metallic state, and then redissolved in a little nitric acid. It may be precipitated on zinc from a sulphuric acid solution in the usual way, but the writer prefers to precipitate it on a few strips of sheet aluminum, from a boiling solution containing 50 c.c. of water and 10 c.c. of strong sulphuric acid. The copper is all thrown down in about five minutes, and may be easily washed, pouring the washings through a filter, and redissolved in nitric acid, which does not attack the aluminum. There appears to be no simple way to remove antimony, but arsenic may be sufficiently got rid of as follows:

Evaporate the original nitric acid solution of the ore, in a small flask, nearly to dryness, and add 5 c.c. of strong hydrochloric acid. Boil till half the hydrochloric acid is gone, and then add about 2 c.c. of a solution of 2 grams of sulphur in 10 c.c. of bromine. Again boil for half a minute, and then add 10 c.c. of strong sulphuric acid, and heat strongly until the latter is boiling freely. As the acids and bromine go off, so does the arsenic. What little may remain will not come down during the subsequent deposition of the copper.

The results obtained by the above apparatus do not differ from those yielded by the ordinary battery method, and the time required does not exceed an hour and a half in any case.

II.—THE CYANIDE ASSAY.

This well-known and rapid method depends upon the power possessed by an aqueous solution of potassium cyanide to decolorize an ammoniacal solution of a copper salt, and is, under proper conditions, quite accurate enough for ordinary mill-work on familiar ores. These conditions are as follows:

(*a*) The use of measured and constant amounts of acid and ammonia.

(*b*) The cooling of the ammoniacal copper solution to nearly the temperature of the surrounding atmosphere before titration.

(*c*) The intimate mixture of the cyanide solution, as it drops from the burette, with the copper solution, and a sufficient, though accurately limited, time in which to accomplish its bleaching action.

(*d*) The establishment of a certain fixed shade of pink at the standardizing of the cyanide solution, to which all subsequent assays must be as closely as possible approximated in color for the finishing point. This renders it impossible for any chemist to work with another person's solution until he has first standardized it himself, and determined its strength according to his own custom.

The absence of any considerable amount of lime, zinc, arsenic and antimony, whose presence has long been known to seriously vitiate results, though exactly to what extent was not demonstrated, until a series of experiments on this point was carried out in 1882 under the direction of the author, and still more recently by Torrey & Eaton.

From a long list of results, some of them even contradictory, the following deductions were drawn:

The presence of zinc in quantities below 4½ per cent. has no perceptible influence on results.

Five per cent. of zinc, in a siliceous ore of copper, containing no other metals except iron, caused a constant error on the plus side of about 0.22 per cent., which increased in a tolérably regular ratio with an increased percentage of zinc.

After eliminating a few results that varied very greatly and unaccountably from all others, an average of about six determinations of each sample yielded the following figures. The ore just described was used in every case, and the zinc added in the shape of a carefully determined sulphide, allowances being also made for the increase in the weight of the ore sample resulting from this addition of foreign matter.

Ore free from zinc,			11.16	per cent.	copper.
No. 1	with 4	per cent. zinc,	11.46	"	"
" 2	" 4½	" "	11.55	"	"
" 3	" 5	" "	11.72	"	"
" 4	" 6	" "	12.1	"	"
" 5	" 8	" "	13.2	"	"
" 6	" 10	" "	13.3	"	"
" 7	" 15	" "	13.9	"	"
" 8	" 20	" "	13.8	"	"

The presence of arsenic and antimony in much smaller proportions—1 per cent. or less—may cause errors on both plus and minus sides to the extent of one-half a per cent. or more, and in larger quantities will generally render the test totally unreliable.

Another indispensable and oft-neglected precaution is the testing of the precipitate of hydrated oxide of iron caused by the addition of ammonia to the original solution. This bulky precipitate, especially in the case of mattes and highly ferruginous ores, will retain a considerable amount of copper which even the most careful washing will not remove, but which may be speedily determined by dissolving the precipitate in the smallest possible quantity of muriatic acid, saturating with ammonia, and again titrating if any blue coloration is produced. The following results, taken from the notebook of a busy chemist, who had never been aware of this possible source of error until accidentally mentioned to him by an assayer in the employ of the writer, and who at once instituted careful experiments to ascertain the probable extent of the mistakes that he had made while acting as assayer to large smelting works, give some idea of the serious discrepancies that may arise from the non-observance of this precaution:

Character of sample.	Without resolution of precipitate			With resolution of precipitate		
No. 1, pyritous ore	21.2	per cent.	copper	23.7	per cent.	copper
" 2, bornite	37.8	"	"	42.4	"	"
" 3, cupola matte	27.7	"	"	31.2	"	"
" 4, reverbatory matte	46.4	"	"	47.4	"	"
" 5, blue metal	57.7	"	"	58.2	"	"
" 6, white metal	74.7	"	"	75.2	"	"
" 7, regule	86.2	"	"	86.5	"	"
" 8, blister copper	97.3	"	"	97.2	"	"

As might be expected, the greatest discrepancies exist in connection with those samples containing the largest amounts of iron, and decrease to nothing as the iron contents diminish.

In the absence of the injurious elements—zinc, arsenic, antimony—the cyanide assay is sufficiently accurate, and, from its simplicity and rapidity of execution, it is peculiarly adapted to the daily working assays from the mine, smelter and concentrator. In fact, it is the mainstay of the overcrowded metallurgical assayer, and can be used for nearly every purpose, except for the buying and selling of ores and copper products, and for the determination of very minute quantities of copper in slags. It is frequently employed with satisfaction for the last-named purpose, a much larger amount than usual being taken, in order to obtain a solution sufficiently rich in copper to exhibit a reasonable degree of color. Messrs. Torrey & Eaton have published additional investigations of great value on the effect of various substances upon the accuracy of the cyanide method. (See *Engineering and Mining Journal*, May 9, 1885.)

In their experiments they employed a cyanide solution capable of showing one-thirtieth of one per cent. of copper, and took every precaution to have all conditions identical during the various tests; all solutions titrated being of the same degree of strength.

Silver and Bismuth.—A solution was made of the following metals:

Copper	.550 gram.
Bismuth	.200 "
Silver	.250 "

The silver was precipitated with hydrochloric acid, and ammonia added after filtering and washing. Two titrations gave:

No. 1	54.90 per cent. copper	
No. 2	54.85 " "	instead of 55 per cent.

These results show that a solution containing the very unusual proportion of 20 per cent. of bismuth and 25 per cent. of silver can be titrated to within 0.1 per cent. of its value in copper.

Lead.—This metal, being a common element in copper ores and alloys, was introduced into a copper solution in the following proportions:

Copper	.200 gram.
Lead	.800 "

After adding ammonia and allowing the lead precipitate to separate for two or three hours, it was titrated, giving 20.28 per cent. of copper, instead of 20 per cent. Messrs. Torrey & Eaton, therefore, believe that the amount of lead commonly present in ores—from 5 to 40 per cent.—would not injuriously affect the operation.

Arsenic.—Torrey & Eaton titrated, without filtering, a solution containing .600 gram arsenic, .400 gram copper, finding 39.8 per cent. instead of 40 per cent. Therefore any ordinary amount of arsenic—from 5 to 15 per cent.—would seem to have no injurious influence.

Ammonia and hydrochloric acid, when indiscriminately used, were found by Messrs. Torrey & Eaton to cause serious errors, the results being influenced to the extent of from ½ to 1 per cent. by any large excess of either.

Lime in large quantity was found to confuse results.
Magnesia had no effect whatever.

LOW'S MODIFIED CYANIDE ASSAY.*

The cyanide method, preceded by the separation of the copper from interfering substances, and more or less modified by different operators, is the one in common use in Colorado for technical work. The writer has adapted a modification of his own, in which the preliminary precipitation of the copper, in the metallic state, is effected by aluminum instead of the customary zinc. The object is to obtain a copper that is unquestionably free from zinc, as it is found that sometimes, when an excess of zinc is employed, some of that metal is retained by the copper, causing too high a result. Of course, such an error is accidental and unnecessary, but it is, nevertheless, sufficiently frequent to indicate the value of a scheme in which it cannot possibly occur. The copper is eventually obtained in a blue, ammoniacal solution, and its amount is estimated from the quantity of standard solution of cyanide of potassium required to discharge the blue color, as in the ordinary cyanide assay. The results of the cyanide titration are exact if certain conditions are always maintained. It is found that for the same amount of copper:

1. A concentrated solution requires more cyanide for decoloration than a dilute solution.
2. A hot solution requires less cyanide than a cold one.
3. In any case when, from a rapid addition of cyanide, the color has become rather faint, it may, by simple standing, continue to fade, and perhaps entirely disappear.
4. If the amount of cyanide added is insufficient to effect complete discharge of color, even after allowing the copper solution to stand for several minutes, the titration may then be finished without alteration of the final result.

From the foregoing facts it is evidently necessary, in order to obtain correct results, that the titrations for unknown amounts of copper should be made under conditions that do not differ materially in the following particulars from those governing the standardization of the cyanide solution:

1. Temperature.

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* Mr. A. H. Low has been kind enough to send me the following description of his very convenient and useful modification of the Cyanide Assay.

2. Rapidity of the final additions of cyanide.
3. Final bulk of solution.

Besides the physical conditions just enumerated, there are chemical conditions that effect the result, such as presence of a large amount of chlorides, a large excess of ammonia, etc. Such abnormal conditions require no special consideration, since they are all easily avoided by following the method to be described.

STANDARDIZATION OF THE CYANIDE SOLUTION.

Dissolve pure cyanide of potassium in distilled water, in the proportion of 21 grams to the liter. The commercial cyanide, dissolved in common water, may be used, but is not recommended. An uncertain quantity, perhaps 60 grams to the liter, is required, and a slimy precipitate or residue is always left, that must either be filtered off, or allowed to settle, so as to decant the pure liquid.

Weigh accurately about 0.200 grams of pure copper foil, and place it in a pear-shaped flask of about 250 c.c. capacity. Add 5 c.c. of strong, pure nitric acid, which will quickly dissolve the copper. Without boiling off the red fumes, add about 80 c.c. of distilled water and 10 c.c. of strong ammonia water (26 degrees Beaumé). Cool to the ordinary temperature, by placing under a tap, or in cold water. Titrate with the cyanide solution, in a slow, cautious manner, and, as the end-point is approached, as shown by the partial fading of the blue color, add distilled water so as to bring the solution to a bulk of approximately 150 c.c. Finish the titration by careful and regular additions of cyanide, finally decreasing to a drop at a time, until the blue tint can no longer be detected by holding the flask against a light-colored background. It is, of course, very essential that there should be no haste and no prolonged delay in these final additions of cyanide. Simply adopt a regular, natural manner, that can easily be repeated on all subsequent titrations. Keep the cyanide solution in a glass-stoppered bottle (the common 2½ liter acid-bottle is convenient), in a cool place not exposed to direct sunlight. Under these circumstances it holds its strength fairly well, but still it gets weaker from the decomposition of the cyanide, and should be restandardized weekly.

TREATMENT OF ORES, ETC.

Treat 1 gram, or 0.5 gram if the material seems to contain 40 per cent. copper or over, in a flask of about 250 c.c. capacity, by boiling first with 10 to 15 c.c. of strong nitric acid to effect decomposition, and then with 10 c.c. of strong sulphuric acid, to expel the nitric acid. In most cases, ores are easily decomposed, and do not require extremely fine pulverization; the assay that has passed an 80-mesh sieve being fine enough. In the case of mattes however, it is best to give an additional grinding, in an agate mortar, of a small portion, from which the sample for analysis is to be weighed.

Boil the contents of the flask gently during the decomposition with nitric acid, and then, after the addition of the sulphuric acid, place the flask over a small, naked flame, and heat until all the nitric acid is expelled, and the residuary sulphuric acid is boiling freely and evolving copious fumes. Remove from the flame and allow to cool. Ores that are not decomposed by this treatment must be attacked in some special manner for which no general directions can be given. Sometimes the addition of hydrochloric acid is all that is necessary. It is advisable in any case not to add the sulphuric acid until the ore appears to be well decomposed.

To the residue in the flask add 50 c.c. of water, and three or more pieces of sheet aluminum, each perhaps $1\frac{1}{2}$ by $\frac{1}{2}$ by $\frac{1}{16}$ inches in size, and heat the mixture to boiling. Boil strongly for at least five minutes, when the copper is ordinarily all precipitated. Remove from the lamp, add 25 c.c. of cold water, allow to settle a moment, and then decant through a three-inch filter, retaining in the flask the aluminum and as much of the copper as possible. Wash the precipitated copper twice by decantation, using about 25 c.c. of water each time, and pouring through the filter. Drain the flask as completely as possible the last time, and then give the filter one or two extra rinsings. Pour into the flask 5 c.c. of strong nitric acid, and shake it about gently until the copper is all dissolved, the aluminum not being attacked. Now add 5 c.c. of water and one drop of strong hydrochloric acid, and, after shaking around for a moment to coagulate any chloride of silver, pour the mixture upon the filter, thus dissolving whatever copper is there, receiving the filtrate in a small beaker. Rinse out the flask, pouring the rinsings through the filter, and remove the aluminum, which is but little attacked, for further use. Finally, wash the filter thoroughly, but with as little water as possible, so as not to obtain too bulky a filtrate, and then transfer the solution back to the flask again. Now add 10 c.c. of strong (26 degrees Beaumé) ammonia water, and cool the solution to the ordinary temperature. Dilute to about 75 c.c., and titrate with the standard cyanide cautiously until the blue color is discharged to a considerable extent, and it is evident that the end point is not far off.

The liquid is now frequently more or less cloudy from the presence of hydrate of lead, and possibly small amounts of the hydrates of iron, aluminum, etc., and for accurate work should be filtered. If the titration has been carried too far before filtration, the faint blue tinge is liable to fade completely away, thus spoiling the assay. On the other hand, it is not advisable to filter the ammoniacal solution before the addition of cyanide, as such a filtered solution will frequently develop a second milkiness during the titration, and have to be filtered again. Filter the partly titrated solution through a 5-inch filter. One washing will usually suffice. Finish the titration very carefully on the clear, pale-blue solution, precisely as in the standardization previously decribed. Toward the end, dilute with water, if necessary, so as to obtain a final bulk of about 150 c.c.

The number of c.c. of cyanide solution required, multiplied by the copper-value of one c.c., gives the weight of copper contained in the amount of ore taken, from which the percentage is readily calculated.

If the amount of silver present in an ore is known, it need not be removed, but may be allowed for, on the basis that $2Ag=Cu$. One per cent. of silver, or 292 ounces per ton of 2,000 pounds, would thus approximately equal 0.293 per cent. of copper; and 100 ounces of silver per ton would equal 0.10 per cent. copper. Accordingly, 0.10 per cent. of copper is to be deducted for every 100 ounces of silver per ton. Thus, if the result of the titration indicates 24.63 per cent. Cu, and the ore assays 250 ounces per ton in silver, the true result for copper is 24.63—0.25, or 24.38 per cent.

None of the ordinary constituents of ores interfere with the method as described. Duplicates should easily agree within 0.10 or 0.2 per cent., which answers for ordinary uncommercial tests.

III.—THE IODIDE ASSSAY.

The following description of the iodide assay, as practiced very largely at English metallurgical works in place of the electrolytic method, has kindly been written for me by Mr. I. H. Clutton, assayer and metallurgical chemist to The Elliott's Metal Company, Limited; Selly Oak Works, Birmingham, England; and Pembrey Copper and Silver Works, Burry Port, South Wales, with the permission of Mr. Gerard B. Elkington.

THE IODIDE COPPER ASSAY.

This assay, which may now be fairly described as the standard English method, is, in practised hands, both accurate and expeditious.

It depends upon the reaction that occurs when an excess of potassium iodide is added to a solution of a cupric salt in a slightly acid, or acetic acid, solution, cuprous iodide being formed and an equivalent of iodine set free. This free iodine is dissolved in the excess of potassium iodide, the liberated iodine being in exact proportion to the copper present. Thus: $2CuSO_4+4KI=Cu_2I_2+2K_2SO_4+I_2$.

The free iodine (*i.e.*, indirectly the copper) is determined in the usual way, by titrating with sodium hyposulphite (thiosulphate), using starch as an indicator. Sodium iodide and tetrathionate are formed.

Thus $2Na_2S_2O_3+I_2=2NaI+Na_2S_4O_6$.

The exact method of conducting the assay is as follows:

From one-half to two grams (100 to 400 milligrams of copper) of the ore or matte, according to richness, is weighed into an 8-ounce flask and dissolved, best by thorough decomposition of the sulphides with nitric acid, and after taking nearly to dryness, by partial evaporation with sulphuric acid; this renders any lead insoluble by converting it into sulphate. The lead sulphate and insoluble residue are then filtered off,* the solution being passed into a 16-ounce flask, in which the copper is precipitated as sulphide either by hyposulphite of soda, or sulphureted hydrogen. The former method has the advantage of not contaminating the atmosphere of the laboratory, while, in making a large number of assays, the latter is more economical, and even perhaps more expeditious. In either case, the sulphides are washed free from iron.

If much iron is present, the solution, during titrating, will have a reddish color, due to ferric acetate. This tends to mask the reaction, but reasonably small quantities of iron do not interfere.

The sulphides are washed back into the flask, and dissolved with 10 c.c. of nitric acid. A little chlorate of potassium may also be

* This filtering may also be omitted.

used at the end, to assist in the liberation of the sulphur. The assay is evaporated on the sandbath to as near dryness as possible; nitrous fumes are blown out, and the copper salts dissolved in a little hot water. The solution is filtered through a small funnel over Swedish filter-paper, into a No. 7 beaker, sulphur and most of the antimony (as oxide) remaining behind, together with lead sulphate and insoluble residue, if the first filtering was omitted.

Instead of dissolving the sulphides direct, some chemists dry, calcine, and dissolve the remaining oxides in a little nitric acid.

The solution, which should not now much exceed 50 c.c. in bulk, is neutralized with sodium carbonate, and a slight excess of acetic acid added.

Potassium iodide crystals are then added, the quantity being immaterial so long as there is enough; about five or six grams will be the proper amount in ordinary cases.

The solution, in which the copper now exists as cuprous iodide, is of a yellow-brown color, and sodium hyposulphite is run into it from a burette, the assay being constantly shaken. The yellow color rapidly lightens, and, on the addition of 5 or 6 c.c. of starch solution, strikes a deep blue color, which disappears on the addition of more hyposulphite, leaving a creamy, white tint, the end reaction being sharp and distinct. The assay is titrated in the cold, to prevent any possible loss of iodine.

The hyposulphite solution alters slightly by keeping, and requires occasional standardizing. Its strength is such that 500 milligrams of copper require from 50 to 60 c.c. (about 40 grams of hypo to one liter of water).

The standards are prepared by dissolving 4 grams of pure electrolytic copper in the smallest possible excess of nitric acid, diluting to one liter, and drawing off 50 c.c. at one time, neutralizing and treating exactly as the assay.

The starch solution is made by pouring a little boiling water on about one gram of starch in a beaker, rubbing the same to a thin paste with a glass rod, neutralizing, and treating exactly as the assays.

Personally, I find that in neutralizing, it is best to have only the slightest excess of carbonate of sodium.

Arsenic does not interfere with the action or affect the result. Bismuth imparts a yellow color to the solution, and the assay is liable to be overdrawn before the addition of starch; otherwise it does not affect the reaction or the result.

Mr. A. H. Low, assayer and analytical chemist, of Denver, Colorado, has been kind enough to furnish me with the following description of certain modifications that he has made in the iodide assay, that tend to render it more convenient and exact.

Mr. Low says: The iodide method for the copper assay, when carried out according to the following modification, devised by the writer, appears to fully equal the electrolytic method in accuracy, and as it requires very much less time, scarcely more than the ordinary cyanide method, it is greatly to be preferred for the usual run of impure ores and furnace products. The following figures are given to show the accuracy of the hyposulphite titration as described: On the basis of one gram as 100 per cent. there were

Taken	0.78	per cent. Cu	....	found	0.79	per cent. Cu.
"	10.30	"	....	"	10.30	"
"	15.37	"	....	"	15.38	"
"	20.31	"	....	"	20.30	"
"	44.45	"	....	"	44.44	"
"	46.92	"	....	"	46.89	"
"	56.82	"	....	"	56.85	"

Can the electrolytic method improve upon this? No special pains were taken with these tests, and they were made as rapidly as the daily technical work. The scheme devised removes all ordinary interfering impurities or renders them inert. Zinc has not been found as good a precipitant for the copper as aluminum for several reasons, one of the principal being that the precipitated copper is frequently contaminated with considerable iron, even when thrown down from strongly acid solutions, and this iron may occasion much subsequent annoyance. When aluminum is used, the precipitation may be effected without boiling by simply adding a few drops of hydrochloric acid to the solution, but this has not been found so desirable as the method described. For the success of the hyposulphite titration it is absolutely essential that there be no nitrate of copper or free nitric acid present. When a solution of nitrate of copper is neutralized in the cold with ammonia, which may even be added in large excess, and the solution is then re-acidified with acetic acid, the mixture behaves toward iodide of potassium as though there were some nitrate of copper or free nitric acid also present. If, however, the ammoniacal liquid be boiled for a moment, the neutralization appears to be complete, the nitric acid all combining with the ammonia and occasioning no subsequent trouble on the addition of acetic acid.

COPPER ASSAY BY THE IODIDE METHOD.

Prepare a solution of hyposulphite of sodium containing about 38 grams of the pure crystals to the liter. Standardize as follows: Weigh accurately about 0.200 grams of pure copper foil and place in a flask of about 250 c.c. capacity. Add about 4 c.c. of strong nitric acid to dissolve the copper and then evaporate down to 1 or 2 c.c., avoiding overheating which might easily convert some of the copper into a basic salt or oxide. The operation may be hastened by manipulating the flask in a holder over a small naked flame. Now add 5 c.c. of water to dissolve the nitrate of copper, and then 5 c.c. of strong ammonia water. See that the copper has all dissolved and the solution is strongly alkaline. Heat to boiling and boil for about a minute. This is absolutely necessary to insure the perfect neutralization of the nitric acid in the nitrate of copper. Remove from the heat and add 6 c.c. of glacial acetic acid, and then 40 c.c. of cold water. Again see that all copper salts are dissolved, and then add to the cool solution about 3 grams of iodide of potassium and shake it about gently until dissolved. Cuprous iodide will be precipitated and iodide liberated according to the following reaction: $2[Cu.\ 2C_2H_3O_2]+4KI=Cu_2I_2+4[K.\ C_2H_3O_2]+2I$. The free iodine colors the mixture brown. Titrate at once with the hyposulphite solution until the brown tinge has become weak, and then add sufficient starch liquor to produce a marked blue coloration. Now continue the titration cautiously until the blue tinge vanishes. Stop at the first decided change, and the color will usually entirely disappear after standing a moment. One c.c. of the hyposulphite solution will be found to correspond to about 0.01 gram of copper. The reaction between the hyposulphite and iodine is: $2[Na_2S_2O_3]+2I=2NaI+Na_2S_4O_6$. Sodium iodide and tetrathionate are formed. The starch liquor may be made by boiling about half a gram of starch with a little water and diluting to about 250 c.c. It should be used cold, and must be prepared frequently for regular work, as it does not keep very well. The hyposulphite solution made of the pure crystals and distilled water appears to be very stable. The writer has never detected any appreciable variation in strength during the time required to use up a lot, say a month or more.

TREATMENT OF ORES.

Treat 1 gram of the ore in a flask of 250 c.c. capacity with 10 c.c. of strong nitric acid by boiling nearly or quite to dryness.

Now add 10 c.c. of strong hydrochloric acid and again boil. After boiling for two or three minutes, add 10 c.c. of strong sulphuric acid and heat strongly, best over a small naked flame, until the more volatile acids are expelled, and the fumes of sulphuric acid are coming off freely. Allow to cool, and then add about 40 c.c. of water and heat to boiling. Filter through a 3-inch filter, more particularly to remove any sulphate of lead, and collect the filtrate in a beaker about three inches in diameter. Wash flask and residue with hot water, and endeavor to keep the volume of the filtrate down to 75 c.c. or less. Place in the bottom of the beaker a piece of sheet aluminum prepared as follows: Cut from stout sheet aluminum a strip about 3 inches long and 1½ inches wide, and bend up each end at right angles for about five-eighths of an inch, or so that the body of the strip will lie flat in the bottom of the beaker. This aluminum may be used repeatedly as it is but little attacked each time. Cover the beaker and heat to boiling. Boil strongly for about six or seven minutes, when the copper will be all precipitated if the bulk of the solution does not exceed 75 c.c. More dilute solutions should be boiled correspondingly longer, as the sulphuric acid does not begin to attack the aluminum strongly until of about the degree of concentration recommended. Now pour the liquid in the beaker back into the original flask, and, with the wash-bottle of hot water, rinse in also as much of the copper as possible, leaving the aluminum behind. The beaker and aluminum, which may still retain some adhering copper, are now temporarily set aside. The copper in the flask is allowed to settle and the clear liquid decanted through a small filter. Wash the copper two or three times by decantation with a little hot water, pouring the washings through the filter but retaining the copper as completely as possible in the flask. Now place the beaker containing the aluminum under the funnel and pour 3 or 4 c.c. of strong nitric acid, drop by drop, over the filter. This dissolves whatever copper may be there and washes it into the beaker. Wash with a little hot water if necessary, but endeavor to keep the total volume of liquid as small as possible. Finally rinse the solution in the beaker into the flask and set the latter over the lamp. As soon as the copper has dissolved, add about half a gram of chlorate of potassium to oxidize any arsenic present to arsenic acid. This is very important. Continue the boiling until only 1 or 2 c.c. of liquid remain, but not so far as to form a basic salt or the oxide of copper. Proceed with the residue precisely as with the residue of nitrate of copper in the standardization of the hyposulphite, finally calculating the percentage of copper present from the amount of standard hyposulphite required.

An excess of iodide of potassium is not necessary. One gram of pure copper requires 5.24 grams of KI, consequently 3 grams of KI are quite sufficient for anything under 50 per cent. Cu, when 1 gram of ore is taken for assay. If the percentage of copper is likely to run above that point, 5 grams of the iodide had better be used. Lead and bismuth form colored iodides, and if present in any considerable amount, mask the end-point before adding starch. They are otherwise without effect, as is also arsenic when oxidized as described. The return of the blue tinge in the liquid by long standing after titration is of no significance.

IV.—THE COLORIMETRIC DETERMINATION OF COPPER.

This is reserved almost exclusively for the determination of minute quantities of copper contained in slags, tailings from concentration, and similar products.

Heine's modification of this method, as described by Kerl, is perhaps the most convenient, and with proper solutions for comparison, preserved in bottles of colorless glass and of exactly the same size, yields results that cannot be surpassed. It is seldom employed for substances containing over one and one-half per cent. of copper, and may be relied upon to show differences of 0.03 of one per cent.; results, however, depending largely upon the skill of the operator, and his capacity for discriminating almost invisible shades of color.

V.—THE LAKE SUPERIOR FIRE ASSAY.*

This fire assay is only adapted to ores free from sulphur and other metalloids. Native copper is the principal substance dealt with, though oxides of copper may be equally well determined by this method. The Lake Superior concentrates consist of metallic copper, and sometimes carry up to 50 per cent. of titanic iron sand. Silica, oxide of iron, and metallic iron from the stamp-heads are also usually present.

* A detailed account, by Mr. M. B. Patch, of this interesting assay was given in the first edition of this work; but subsequently, owing to the rapid accumulation of material that is of more value to the majority of copper metallurgists, I have reluctantly felt obliged to omit it.

Sodium bicarbonate, borax, potassium bitartrate, ferric oxide, sand, and slag from the same operation are the chief fluxes employed.

The fluxed sample is fused in a wind furnace for about 25 minutes, the resulting button being almost pure copper, and its weight agreeing very closely with battery assays of the same sample.

It is needless to say that very much depends upon the skill and experience of the operator.

The same assay is used for the determination of copper in slags carrying half a per cent. of copper, and less.

I append a table, showing the fluxing-formulas for various samples of concentrates. Also tables showing results of this assay as compared with electrolysis.

(These tables are from Mr. Patch's valuable paper, as is indeed, the above brief abstract.)

Mineral.		Weight. Grains.	Borax. Grains	Soda. Grains.	Slag. Grains.	Potassium Bitartrate. Grains.	Sand. Grains.	Iron Ore. Grains.
Number.	Copper. Per cent.							
1.......	92	1,000	60	55	200	300	......	....
2.......	86	1,000	60	60	180	300	......	....
3.......	60	500	100	80	... ..	300	......	....
4.......	33	500	150	160	......	300	150	....
5.......	20	500	190	200	......	300	175	....
*.......	35	500	140	140	......	300	......	100
†... ...	5 to 20	500	200	200	......	300	......	....

The results obtained by this method are surprisingly accurate. Duplicate determinations of the lower grade samples seldom vary more than 0.1 or 0.2. A difference of 0.4 per cent. is a rare occurrence, even in the higher classes of mineral, where the size of the metallic fragments renders the sampling, and even the weighing out of a correct assay a matter of some uncertainty.

A few results from Mr. Patch's notes will confirm these statements. An average series of tests on cupola slags by the colorimetric method for the period of a month, duplicated by the fire assay, gave a result 0.05 per cent. lower for the latter test, the slag containing about 0.5 of one per cent.

As an illustration of the results of this system when applied to very rich ore, a comparative test was made for eight days on No. 1 Calumet & Hecla mineral, with the following results:

Battery assay..................................	89.100 per cent.
Fire assay.....................................	88.812 "

A similar test on No. 2 Calumet & Hecla mineral:

Battery assay......	77.590 per cent.
Fire assay.............	77.657 "

A similar test with various samples:

No.	Battery Assay.	Fire Assay.	
1..................	Mean — 89.544	89.50	Mean — 89.92
2..................		89.60	
3..................		89.70	
4..................		89.70	
5..................	Mean — 77.740	77.40	Mean — 77.50
6..................		77.50	
7..................		77.70	
8..................		77.40	

It is a somewhat curious fact that the slight loss of about 0.25 per cent. of copper, which results from the passage of a minute portion of the metal into the slag, is just about counter-balanced by the impurities in the copper button from the reduction of ferric oxide, the amount of which is indicated by the following analysis of copper buttons—the only weighable impurity being iron:

Copper. Per cent.	Copper. Per cent.	Copper. Per cent.
99.83	99.76	99.51
99.84	99.80	99.87
99.52	99.46	99.79

This account of a little known process will doubtless remove the impression sometimes held by chemists that the Lake Superior copper assay is a clumsy and imperfect operation, and unworthy any advanced system of metallurgy.

THE DETERMINATION OF GOLD AND SILVER IN COPPER FURNACE MATERIAL.

The presence of a large proportion of copper demands special precautions in assaying matte or bars for the precious metals.

The following methods will be found simple and accurate, and are those usually employed by public assayers, and at the principal smelters.*

We notice in the outset a divergence between the methods usually employed in the east and west of the United States. Most

* This description is taken from a paper read by Mr. A. H. Ledoux before the American Institute of Mining Engineers, October, 1894, entitled "A Uniform Method for the Assay of Copper Furnace Materials for Gold and Silver."

of the Eastern public assayers, as well as those employed by Eastern smelting works, use what may be called a wet method, but is, strictly speaking, a combination method of assay. While there are many details incidental to different laboratories, this wet method may be outlined briefly as follows:

For Gold—One assay-ton of the copper-borings or matte is transferred to a No. 5 beaker with a clock-glass cover. The sample is treated with a mixture of 100 c.c. of water and 50 c.c. of nitric acid of sp. gr. 1.42. When the violent action has ceased, 50 c.c. more of the nitric acid is added, and the solution is gently heated until everything soluble has been dissolved. The contents of the beaker are then raised to the boiling point, the cover is removed, and boiling is continued until most of the nitric acid has been expelled. The solution is then diluted with about 400 c.c. of water free from chlorine, 5 c.c. of concentrated sulphuric acid is added, and then 10 c.c. of a concentrated solution of either acetate or nitrate of lead. The dense white precipitate of lead sulphate carries down with it the minute particles of gold which may be suspended in the solution.

The precipitate is then allowed to settle for some hours—over night, if possible. It is then filtered, washed once or twice with water, the beaker is carefully cleaned, and the filter and contents, now practically free from copper, are partially dried, wrapped in thin lead-foil, and transferred to scorifiers; enough test-lead is added to bring the total lead present up to 50 grams, a pinch of borax glass is placed on top, and the scorification is conducted as usual. It is necessary to raise the temperature gradually until the paper has been consumed and the contents of the scorifier melted down. Cupellation is conducted in the usual manner.

This method is intended for the determination of gold; but enough silver will be present to allow the bead to be parted. When, however, considerable gold, say two or three ounces per ton (0.01%,) is supposed to be present, it is well to add a drop of salt solution to the original nitric acid solution, to precipitate some of the silver along with the lead, or else to add a small amount of pure silver at the time of scorification. It is important not to precipitate all the silver, as in that case there might be an excess of salt which might liberate chlorine and vitiate the results as to gold.

For Silver.—The usual method employed in the East for the assay of copper bars, mattes, ores, etc., containing silver is likewise modified in different laboratories. These modifications vary, as a rule, with the supposed richness in silver of the sample treated The sample is dissolved in dilute nitric acid, as described in the above method for gold. To the solution, after the addition of sulphuric acid and before that of lead acetate, a solution of chloride of sodium is added in a sufficient quantity to throw down all the silver, the addition being gradual, and avoiding a great excess (as silver chloride is more or less soluble in sodium chloride solution); then the lead acetate is added, the solution is well stirred, and the mixed precipitate of lead sulphate and silver chloride is allowed to settle as in the gold determination. The rest of the process is conducted exactly as in the previous case for gold. Where any considerable amount of gold is present it is of course necessary to part the beads and deduct the weight of gold present, which otherwise would be weighed as silver, thus erroneously increasing the proportion of this metal. The gold obtained by this parting is usually less than the figures obtained by the special assay for gold, because some of the gold is dissolved by chlorine through the excess of sodium chloride employed.

Some assayers determine the gold and silver at one operation by taking the filtrates from the gold and lead sulphate precipitate obtained as above described, precipitating the silver in this solution as chloride, adding more lead acetate, and after filtering, combining the two filter papers, one containing the gold and the other the silver, and uniting them for one scorification and subsequent cupellation. This method is more economical for the assayer, and has the advantage also of two filtrations for gold, catching any fine particles which might pass through the first filter; but on the other hand it takes more time, because the same solution is twice settled. In the first method, the settling of the gold and silver precipitates goes on simultaneously.

In the West, the all-fire method is employed almost exclusively, so far as I can ascertain. In the Omaha and Grant works, for example, ten portions of sample, of one-tenth A. T. each, are weighed out and scorified with 50 grams of test-lead, one-half of which lead is mixed with the sample and the remainder used to cover it in the scorifier. One gram of borax is added. The lead buttons obtained by the scorification are cupelled separately, but the ten beads are weighed together. The cupels are then ground up and fused in five lots of two each, with the following charge: Litharge, 90; soda, 50; borax-glass, 50, and argols, 3 grams. The five buttons are cupelled and the silver is added to that obtained in the first operation, representing the loss in scorification. All the beads are then parted for gold, which is deducted from the total weight as usual.

My experience shows that the determination of gold obtained by this process is usually higher than where the wet process previously described is employed. It may be well to give certain instances in my own experience. On high-grade copper bullion, which contains on an average about 400 ounces of silver per ton (1.37%), the results were:

	Fire Assay.	Combination Wet and Dry Assay.
Gold, ounces per ton......	1.06 [0.00364 per cent.]	0.92 [0.00316 per cent.]
" "	1.32 [0.00453 per cent.]	1.24 [0.00426 per cent.]
" "	0.34 [0.00117 per cent.]	0.20 [0.00069 per cent.]

In bullion containing 300 ounces of silver per ton (1.03%):

	Fire Assay.	Combination Wet and Dry Assay.
Gold, ounces per ton......	4.06 [0.01395 per cent.]	3.96 [0.0136 per cent.]
" "	2.76 [0.00948 per cent.]	2.56 [0.00879 per cent.]
" "	2.72 [0.00934 per cent.]	2.44 [0.00838 per cent.]

In matte containing 60 per cent. of copper and 60 ounces of silver:

	Fire Assay.	Combination. Wet and Dry Assay.
Gold, ounces per ton......	0.24 [0.00082 per cent.]	0.20 [0.00069 per cent.]

The two processes usually agree very closely for silver, provided the cupel-absorption is determined when the silver is assayed by the combination wet process. This cupel-absorption is very much less by the wet process than by the all-fire method, because by the former the copper has been eliminated, and is not present to help carry the silver into the cupel. In some instances, where substances are present which would cause volatilization of silver in scorification, the wet assay gives higher figures, because the interfering substance has been removed by the acid.

The Western all-fire process for mattes is similar to that employed for bars, except that a second scorification is sometimes necessary before cupellation. The second scorification is usually performed in a small 2½-inch scorifier, enough test-lead being added to the button obtained from the first scorification to make the lead present not less than 35 grams.

The above descriptions, as will readily be seen, are in the baldest outline; and it must not be inferred by those interested that all precautions are not adopted to make the results correct; such, for instance, as igniting and dissolving any sulphur-balls which may form when the matte or sulphuret-ores are dissolved in acid, and adding the product to the main solution before precipitating the silver with lead. This precaution is hardly necessary, however, as the very small amount of matte or ore held by the sulphur would be decomposed in the scorification.

Each of these methods in the hands of assayers skilled in its application will produce very uniform results; and yet, as will be seen from the few comparisons given above, any assayer running the two, side by side, will get divergent figures for gold.

A METHOD FOR DETERMINING SULPHUR IN ROASTED SULPHIDE ORES.*

The following method for the estimation of sulphur in materials containing it in the form of sulphides not decomposable by hydrochloric acid is found, in practice, to be exceedingly accurate and convenient. The great majority of methods now practised consists in oxidizing—either in the dry way or wet way—the sulphur to sulphuric acid, and estimating the latter gravimetrically or by titration.

The method about to be described is likewise based on this principle, and is a combination of well-known reactions. What is claimed for it is that it is especially adapted to the quick determination of the sulphur in roasted copper ores and cupriferous pyrites.

The conversion of the sulphur into an alkaline sulphate is effected by fusion with potassium hydrate and sodium peroxide, and the amount of sulphur is then ascertained in the usual manner—gravimetrically, when accuracy is the principal object—by the Wildenstein method, when rapidity is aimed at.

The details of the process are as follows: Five to six grams of caustic potash (pure by alcohol) are fused in a nickel crucible and heated until the excess of water is expelled. The size of the flame is now reduced so that the contents of the crucible just remain liquid, and 0.5 grams of the finely powdered material introduced in small portions. A gram of sodium peroxide is then added while the heat is gradually increased to redness, and this is maintained for a few minutes. After cooling, the fused mass is dissolved in

*This method was devised by Harry F. Keller and Philip Maas, and communicated to the Franklin Institute, January 18, 1895.

water and the solution filtered with the aid of a pump. The undissolved residue is washed four or five times with hot water.

The colorless filtrate is acidified with hydrochloric acid (8 to 9 c.c., sp. gr. 1.2) and boiled to expel carbonic acid. (Before filtering, the liquid is often colored purple by a small amount of ferrate of potassium; a blue color in the filtrate indicates that too much potash was used.)

If the estimation is to be made gravimetrically, the sulphate of barium is precipitated from the boiling liquid in the usual manner. In case, however, titration is resorted to, the liquid is made alkaline with ammonia (about 5 c.c., sp. gr. 0.9). A slight excess of barium chloride solution is added from a burette, and the excess measured with an equivalent solution of bichromate of potassium. A distinct yellow color of the liquid marks the end of the reaction. After a little practice it is generally easy to strike this point, though it will sometime happens that the precipitate does not settle rapidly. In such doubtful cases portions of the liquid should be filtered off. Care should also be taken that the liquid does not become too dilute. It is convenient to prepare the standard solution of such strength that 1 c.c. equals 0.005 grams of sulphur, *i.e.*, indicates 1 per cent. in a sample weighing 0.5 grams.

The solution of barium chloride is prepared by dissolving 38.109 grams of the crystallized salt to a liter, while the bichromate solution should contain 23 grams of the salt per liter.

To test the accuracy of this method a considerable number of determinations were made of the sulphur in a typical roasted copper ore from Montana.

By oxidation with nitric acid and with aqua regia the percentage of sulphur in this material had been found to be 7.095 per cent. and 7.14 per cent. respectively.

Somewhat lower results were obtained by fusion with caustic potash and potassium chlorate, a method which had been used by one of us to control the workings of a lead-ore roasting furnace. The figures varied from 6.78 per cent. to 6.92 per cent.

Our first attempts to oxidize the ore by means of sodium peroxide were not successful. By using 10 grams of potash and 3 to 5 grams of peroxide, figures much lower than those given before resulted. The oxidation was evidently incomplete. When bromine water was added to the solution of the fused mass, 6.82 per cent. of sulphur were obtained.

To our surprise, a higher percentage was also found when less of the peroxide was employed. Thus with 10 grams of potash and 1 gram of peroxide, the determinations averaged 6.8 per cent. The large excess of alkali employed in these fusions invariably caused the solution of some copper, which renders titration impossible. Our next step, therefore, was to reduce the amount of potash.

When 5 grams of hydrate and 1 gram of peroxide were taken, the filtered solution of the fused mass was entirely free from the blue tint produced by the copper, and it is seen from the following figures that the oxidation of the sulphur was complete:

1	6.71	per cent.	sulphur.
2	6.82	"	"
3	6.79	"	"
Average	6.77	"	"

Volumetric estimations gave the following results:

1	6.74	per cent.	sulphur.
2	6.86	"	"
3	6.89	"	"
4	7.14	"	"
5	6.97	"	"
Average	6.92	"	"

Another series of determinations, in which a preparation of potash marked *puriss pro analys** was used, yielded:

1	6.70	per cent.	sulphur.
2	7.09	"	"
3	6.71	"	"
4	6.85	"	"
5	6.79	"	"
6	7.00	"	"
Average	6.85	"	"

A final series, in which the dircetions given in this paper were strictly adhered to, resulted as follows:

*A correction of 0.35 per cent. was necessary in this case, the potash being less free from sulphur than that labeled " pure by alcohol."

1	6.9	per cent.	sulphur.
2	6.75	"	"
3	7.15	"	"
4	7.05	"	"
5	7.05	"	"
6	7.10	"	"
7	7.14	"	"
Average	7.05	"	"

The time required for the volumetric assay does not exceed thirty minutes.

CHAPTER IV.

THE CHEMISTRY OF THE CALCINING PROCESS.

*Roasting or calcination, used indiscriminately in the language of the American copper smelter, signifies the exposure of ores of metals containing sulphur, arsenic, and other metalloids, to a comparatively moderate temperature, with the purpose of effecting certain chemical, and rarely mechanical, changes required for their subsequent treatment. This definition is restricted to the dry metallurgy of copper, and does not take into consideration chloridizing roasting, roasting with sulphate of soda, and other well-known variations, which belong either to the metallurgy of the precious metals or to the wet treatment of copper ores.

The care and attention which should be devoted to this preparatory process cannot be too strongly insisted on, nor can any one carry out either this apparently simple roasting or the following fusion to the best advantage, who is not thoroughly familiar with the striking chemical changes that in every calcination follow closely upon each other, and by which the sulphides and arsenides of the metals are transformed at will into a succession of subsulphides, sulphates, subsulphates, and oxides. These, reacting upon each other according to fixed and well-known laws, enable the metallurgist at his pleasure to produce every grade of metal from black copper to a low-grade matte that shall contain nearly all the metallic contents of the ore in combination with sulphur. To avoid constant repetition, it may be understood that in speaking of calcination, when sulphur is mentioned, its more or less constant satellites, arsenic and antimony, are also included, their behavior being somewhat similar under ordinary circumstances. These very different products, as well as the amount of ferrous oxide, the most important basic element of every copper slag, result solely from

* In English metallurgical literature, the term roasting is applied exclusively to that process in which copper matte in large fragments is exposed on the hearth of a reverberatory furnace to an oxidizing atmosphere, and a moderate, but gradually increasing, temperature.

the degree to which the calcination is carried. In fact, it may be taken as literally true, that the composition of both the valuable and waste products of the fusion of any sulphide ore of copper is determined irrevocably and entirely in the roasting-furnace or stall. A more thorough study of the reactions just referred to will be found in its proper place. Enough has here been said not only to explain the author's object in devoting so much attention to this process, but also to induce such smelters as are not already thoroughly familiar with the theory of calcination to endeavor to become so if they desire to ever excel in the economical treatment of sulphide ores.

The varieties of calcination, as applied to the dry treatment of copper ores, are at most two:

1 The oxidizing roasting, which is necessarily combined with volatilization.

2. The reducing roasting, limited in its application almost exclusively to substances containing much antimony or arsenic.

Plattner's admirable work on *Röstprocesse* contains the whole theoretical part of calcination; but a foreign language is a barrier to many ardent students of metallurgy, and his descriptions and plans of furnaces and apparatus apply to those in use during the past generation. A modern treatise on roasting, regarding the subject principally from a practical standpoint, and adapted to present American conditions, seems desirable. Such a treatise, however, could not attain the highest degree of usefulness without a consideration of the theory of calcination sufficient to enable and encourage all who make use of the more practical part to follow with ease the chemical reactions on which the process is based.

A sufficient idea of the chemical reactions that occur in this important metallurgical process may be obtained by following an ordinary pyritous ore in its passage through the roasting-furnace, and carefully noting all the changes that it undergoes from the moment of its introduction until it is ready for the succeeding fusion; nor are the conditions in either roast-heaps or stalls so different as to require any separate consideration.

A typical ore for this purpose might consist of a large proportion of pyrite, say 45 per cent., some 20 per cent. of chalcopyrite (containing about one-third copper), with a slight admixture of zinc-blende, galena, and sulphide of silver, while the remainder of the ore would usually consist of quartz or silicious material, which may be regarded as practically inert in its effect upon the process of calcination. A charge of such ore, being introduced upon the hearth of a roasting-furnace still at a bright red heat from the preceding operation, exerts a powerfully cooling influence upon the glowing brick-work, and within ten or fifteen minutes reduces the temperature to a point below the ignition point of sulphur, the ore at the same time giving off its moisture, and gaining so much heat that a very slight aid from the fuel on the grate is sufficient to start the oxidation of the iron pyrites, as shown by the blue, flickering flame that plays over the surface of the charge, beginning at that portion of the same that borders on the already hot charge occupying the adjoining hearth, and gradually advancing toward the rear, until every square inch of surface is in a state of active combustion. The rapidity of this process of oxidation varies according to the degree of temperature and the sharpness of the draught, but should not occupy more than an hour from the first introduction of the charge. The composition of iron pyrites (FeS_2) is such that, while one atom of sulphur is united to the iron with considerable tenacity, the second atom is held by very feeble bonds, and becoming volatile at the moderate temperature of the calcining furnace, unites with the oxygen of the air, forming sulphurous acid (SO_2), which escapes in the form of an invisible gas. This reaction is accompanied by a very considerable evolution of heat and the flickering blue flame already mentioned. Being entirely dependent upon the oxygen derived from the air, this reaction is confined principally to the surface of the charge, which, if left undisturbed, would soon undergo a slight fusion, causing a caking of the ore, and still further hindering the extension of the process. It is therefore just at this point that the necessity for frequent and vigorous stirring becomes strikingly apparent. By this manipulation, any incipient crust that may have formed is broken up, the temperature of the layer of ore is equalized throughout its entire depth, and fresh particles of ore are constantly exposed to the influence of the air.

The stirring should begin on the first appearance of the blue flame, and continue for ten minutes at a time, with equal intervals of rest, during which time the working openings should be closed, while an ample air supply is admitted through the regular channels provided for this purpose. The stirring should take place from both sides of the furnace at the same time, and should be systematic, vigorous, and thorough; extending to the very bottom of the charge, and omitting no portion of the ore.

During this period of roasting, and until the disappearance of the blue flame, the roast gases consist almost exclusively of sulphurous acid, together with steam from the moisture present, and the invariable products of the combustion of the fuel.

It will, of course, be understood that the SO_2, and other roast gases, form but a small proportion—seldom more than 2 per cent.—of the air issuing from a calciner stack; atmospheric air always being present in overwhelming proportions. The SO_2 results from the direct oxidation of one atom of the sulphur contents of the iron pyrites, or, when the temperature is somewhat high, of the absolute volatilization of this atom of sulphur as sulphur, and its immediate combustion to SO_2.

The next stage of the process may be reckoned from the beginning of the oxidation of the iron of the pyrites, and also of its second atom of sulphur. This is a much less rapid and vigorous process than the preceding, and is attended by the formation of a certain amount of sulphuric acid, in addition to the sulphurous acid, which is still generated in large quantities. The means by which the former acid was produced was not clearly understood until Plattner's patient and ingenious researches developed the "contact theory," according to which sulphurous acid and the oxygen of the air, in the presence of large quantities of heated quartz, or other neutral material, combine to form sulphuric acid, which may escape invisible, or in the form of white vapors when hydrated, or may in the instant of its formation combine with any strong base that may be present.

In the case under consideration, protoxide of iron (FeO), arising perhaps from the very particle of pyrites whose oxidation gave rise to the sulphuric acid, is at hand; and while the greater proportion of the sulphuric acid formed escapes into the atmosphere, a certain amount combines with the protoxide of iron to form ferrous sulphate, whose presence may easily be detected, owing to its solubility in water.

From the very commencement of the formation of sulphuric acid, a new and powerful oxidizing agent is gained, as the protosulphate of iron is easily broken up by heat. The decomposition of its acid into SO_2 and O promotes the oxidation of other sulphides present to sulphates, while the protoxide of iron is raised to the sesquioxide of that metal—a tolerably stable compound, and one usually found in large quantities in thoroughly roasted pyritic ores. Before the complete decomposition of the ferrous sulphate has occurred, and indeed while some considerable proportion of sulphide of iron may yet remain, an analogous process takes place with the chalcopyrite, its ferruginous portion following almost precisely the same course as the iron pyrites, while its copper contents are transformed into cupric sulphate, which, on the addition of water, becomes copper vitriol, easily recognized by its color and by several simple and well-known tests.

As the process continues, and the temperature is gradually raised, this salt also undergoes decomposition, yielding at first a basic sulphate of copper, which, upon losing its acid, becomes a dioxide and eventually a protoxide of that metal. These last changes, however, require a protracted high temperature.

The oxidation of the iron present is pretty well advanced at the time of the maximum formation of cupric sulphate; but it is not until the decomposition of at least 75 per cent. of the last-named salt that the formation of sulphate of silver begins with any considerable energy. When once fairly started, however, this interesting and important reaction progresses with great rapidity, and the decomposition of the comparatively large proportion of sulphate of copper present furnishes ample oxidizing influence for the minute quantities of sulphide of silver. The maximum formation of the latter substance usually coincides with the almost entire destruction of the former salt, and it is at this point that the Ziervogel calcination should terminate, as any further exposure of the silver salt to heat lessens its solubility in water, and may even threaten its existence. The complete decomposition of the argentic sulphate is only accomplished by a long exposure to a high temperature, which is now easily borne by most ores and mattes, the easily melted sulphides having been converted into almost infusible oxides and basic sulphates.

Galena (sulphide of lead), when present, is converted almost entirely into a sulphate of that metal, which, by a higher temperature, is partially decomposed with the evolution of sulphurous acid and the final production of a mixture of free oxide of lead with sulphate, the proportions of these two substances varying according to the quantity of foreign sulphides present.

Zinc-blende requires a higher heat for its thorough oxidation than any of the preceding sulphides, but with care may be eventually changed into an oxide, although a certain amount of basic sulphate of zinc nearly always remains. This includes all the sulphides assumed to have been present in the ore under consideration, nor will others be encountered in practice unless under very exceptional circumstances. Sulphide of manganese is an occasional unimportant constituent of mattes, and presents no particular difficulty in calcining, being easily oxidized to a basic sulphate, insoluble in water, which is stable except at the highest roasting temperatures, when it yields up its acid in the shape of SO_3, and remains as a mixture of manganous and manganic oxides.

The gangue-rock of copper ores, being usually silicious, undergoes no change and exerts no influence upon the calcining process, except in so far as it assists in the oxidation of sulphurous to sulphuric acid by *contact*, as already mentioned.

Calc-spar loses its carbonic acid and is converted into gypsum (calcium sulphate), while heavy spar—sulphate of baryta—undergoes no change, except in the presence of a powerful reducing atmosphere and at a high temperature, when it may be changed into sulphide of barium. This is soluble in water, and it has been suggested to use its solubility to remove it when its presence is particularly objectionable. A number of trials in this direction were made by the author in 1872 on the heavy spar ores of Mount Lincoln, Colorado, with very poor results; as it was found extremely difficult to reduce the barium sulphate to sulphide without mixing an amount of coal-dust with the ore at least equal to the weight of the heavy spar present—from 30 to 40 per cent.—while the BaS formed at this high temperature is only partially soluble in water.

Arsenic and antimony, when present, are usually combined with some metallic base, and behave like sulphur to a certain extent; but they give off a much smaller proportion as volatile antimonious and arsenious acids, while they combine to a much greater extent with the metallic bases, forming salts difficult to decompose and extremely injurious to the quality of the copper.

Under such circumstances the roasting should be continued in the usual manner until all the sulphides present are oxidized and the resulting sulphates for the most part decomposed. At this stage, from 4 to 6 per cent. of charcoal dust, or fine bituminous or anthracite coal-screenings, should be thrown upon the charge and thoroughly incorporated with it by vigorous stirring, the heat at the same time being raised to the highest practicable limits. The antimoniates and arseniates of iron and copper are rapidly reduced by this means, and a considerable proportion of the injurious metalloids is volatilized, much to the benefit of the resulting copper. The charge should remain in the furnace until all the incorporated carbon is consumed.

In the foregoing description the process of calcination has been carried much further than is generally needed, or even desired, in an ordinary oxidizing-roasting as a preliminary to fusion.

Sufficient sulphur must always be present in the smelting mixture to prevent the formation of too rich a matte, which entails heavy losses in metal, and other injurious consequences. But it is not a simple matter to determine in advance exactly the amount of sulphur necessary to produce a matte of any given grade. This depends not only upon the character of the furnace process to be employed—that is, whether blast or reverberatory—but also to a considerable extent upon the manner in which the residual sulphur is combined with the bases present; the rapidity of the fusion; the quality of the fuel; the volume and pressure of the blast; the character of the gangue and flux; and numerous other factors. Whatever may be the condition of affairs, however, it may be pretty safely predicted that the percentage of the resulting matte in copper will almost invariably be very considerably lower than is either expected or desired, so that there is little danger that the calcining department of any newly constructed plan will have too great a capacity in proportion to the rest of the establishment, and many serious errors and disappointments can be traced directly to this habit of over-estimating the probable quality of the matte and failing to provide sufficient calcining appliances.

In case of calcination previous to smelting in reverberatories, it is well to avoid an excess of air toward the close of the roasting process—a precaution easily effected by closing the working openings as far as possible, the rabble passing through a hole in the center of a divided door, while the passage of any considerable proportion of undecomposed air through the grate is rendered unlikely by the lively fire that belongs to this period. By these precautions the oxidation of any large proportion of the iron present to a sesquioxide is prevented, the latter being infusible and unfit to enter the slag until it is reduced to a protoxide. This reduction takes places instantaneously in the powerful carbonic-oxide atmosphere that prevails in the blast furnace; but in the almost neutral atmosphere of the ordinary reverberatory the sulphur alone plays the part of a reducing agent, and a charge composed of the sesquioxide of iron will be found materially to delay the process of fusion, besides producing a thick and foul scoria. The natural remedy is the admixture of a few per cent. of fine coal stirred thoroughly into the mass of the ore, and fired on vigorously.

Some kind of an idea may be obtained of the probable composition of the matte to be produced at any given time by the ordinary "matte fusion assay," as given in all works on assaying, wherein the ore to be tested is rapidly melted with merely enough borax and silicious flux—say, 100 per cent. of borax and an equal amount of pulverized window-glass—to flux its earthy constitutents, some 10 per cent. of argols,or other reducing agents,being also added.

But the results are far from satisfactory, and after patiently using it for some two years, and being oftener misled than guided by its results, I discarded it completely, and trusted principally to the eye, occasionally aided by the following calculation, which gives better results than any other familiar to me:

Taking the contents of copper in the charge as a standard for comparison, sufficient sulphur should be allotted to it to form a subsulphide, the excess of sulphur still remaining being supplied with sufficient iron to form a monosulphide of that metal. If other metals are present, such as lead, zinc, or manganese, three-fourths of the former, one-half of the second, or one-fourth of the latter substance may be first considered as forming a monosulphide with the sulphur, there being in such a case much less of the metalloid left to take up iron. This rule gives quite accurate results in rapid blast-furnace smelting, and where abundance of iron is present. If the rate of smelting be slow, and considerable lime or magnesia be present, 5 per cent. of the sulphur contents of the charge should be deducted before beginning the calculation; and if the smelting furnace is a reverberatory, the resulting matte will average 8 per cent. higher in copper than is found by this formula.

A simple illustration will make this method of calculation more clear.

We will assume that a roasted ore having the following composition is to be smelted in a blast-furnace:

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* This calculation refers entirely to the older method of smelting, without attempting any oxidizing action in the blast furnace.

ANALYSIS OF CALCINED ORE.

Cu	— 9.0 per cent.	Pb	— 2.0 per cent.
Fe	— 45.0 "	S*	— 7.8 "
SiO_2	— 27.0 "	O and loss	— 7.2 "
Zn	— 2.0 "		
		Total,	100.00

CALCULATION OF MATTE WHICH SHOULD RESULT FROM FUSION OF THE CALCINED ORE.

Following the rule given,

9 Cu require 2.27 S to form a subsulphide.
¾ of 2 Pb require 0.23 S to form a sulphide.
½ of 2 Zn require 0.50 S to form a sulphide.

This provides for 3 per cent. of the 7.8 per cent. of sulphur present, leaving 4.8 per cent., which will take up enough Fe to form a monosulphide. Calculation shows that 8.4 per cent. of Fe will thus be required, leaving 36.6 per cent. available for the slag.

In order to express the composition of the matte just calculated, in the ordinary manner, we multiply the amount of each ingredient by a common factor that will reduce it to a percentage. In this case the factor is 3.61.

9 Cu + 2.27 S	— 11.27 × 3.61	— 40.69	per cent.	Cu_2S.
1.5 Pb + 0.23 S	— 1.73 × 3.61	— 6.25	"	PbS.
1.Zn + 0.50 S	— 1.5 × 3.61	— 5.41	"	ZnS.
8.4 Fe + 4.80 S	— 13.2 × 3.61	— 47.65	"	FeS.
7.8 per cent. S		100.00	"	

Thus the matte from such a charge will contain about 32.5 per cent. copper; the slight loss of sulphur by volatilization and as SO being usually fully balanced by the presence in the matte of a certain proportion of subsulphides in place of sulphides, or even of metallic iron.

The same charge smelted in a reverberatory furnace would yield a matte of about 40 per cent. Cu.

The proper composition of the slag has not been particularly considered in this example. It would be somewhat too siliceous for blast-furnace work, requiring the addition of a little limestone; while for reverberatory work it would be about right as it stands.

From the foregoing statements it is evident that in ordinary copper smelting the calcination of sulphide ores need seldom be

*As most of the oxidized compounds of sulphur contained in the calcined ore will be reduced to sulphides in the cupola furnace, it is proper to estimate all the sulphur present as metallic sulphur.

pushed to the point of perfection indicated when treating of the chemical reactions that take place in the roasting. On the contrary, a due regard for the proper quality of the resulting matte and slag will probably render it advisable to stop the calcining process long before the decomposition of the sulphate of copper in the charge is complete, and even while a considerable portion of undecomposed sulphides still remains. If, however, the calcination has been carried too far, it is very easy to regulate matters by the addition to the smelting mixture of a small proportion of raw sulphuret ore.

A glance at the behavior of the various compounds of sulphur and bases is essential for the clear understanding of the much greater richness of the matte resulting from the fusion of any given charge in a reverberatory than in a blast-furnace, and of the importance of having a certain proportion of sulphates and other oxidized compounds in the smelting mixture, in order that they may react on each other in the manner best calculated to eliminate the residual sulphur, and thus in a measure make up for imperfect roasting.

In the blast-furnace but little sulphur can be directly volatilized, and, consequently, simply fuses with the copper or iron present to form the artificial sulphide called matte. But the sulphates in the presence of carbonic oxide may undergo the following reaction: $CO+FeO, SO_3=CO_2+SO_2+FeO$; the carbonic oxide burning to acid, while the sulphuric acid is reduced to sulphurous acid, which escapes by volatilization, and the protoxide of iron unites with silica to form a slag. But this is true of only a very small proportion of the sulphates present, as in the powerful reducing atmosphere of the blast-furnace, the sulphurous acid, even when once formed, comes in contact with an overwhelming proportion of CO, which in burning to CO_2 robs the SO_2 of its oxygen, reducing it to sulphur, in which condition it unites with iron or copper and enters the matte, thus increasing the amount of this product, while it robs the slag of its most valuable constituent. It is interesting to note the striking difference of the reaction in the reverberatory furnace, where the atmosphere may be regarded as neutral; CO, the most powerful reducing agent, being virtually wanting:

$$Cu_2S + 4\,CuO, So_3 = 6\,CuO + 5\,SO_2.$$
$$Cu_2S + 2\,CuO, SO_3 = 2\,Cu_2O + 3\,So_2.$$
$$Cu_2S + 2\,Cu_2O = 6\,Cu + SO_2.$$

By studying these formulæ, it will no longer seem strange that the reverberatory produces so much richer matte than the blast-furnace from the same charge. Nearly all the reactions between sulphides and sulphates result in the formation of oxides and volatile SO_2, and were it not for an almost invariable preponderance of undecomposed sulphides in the charge, the elimination of the sulphur might theoretically be almost complete. It is by this all-important, but frequently neglected, establishment of a proper proportion between the sulphides and sulphates, that extraordinary results may be obtained in reverberatory smelting, and the roasting plant greatly reduced, as shown in chapter on "Direct Method of Refining Copper."

Although treating of smelting, this matter belongs strictly to the calcining department, and presents a field for study of great interest and practical value. A close analogy may be found in the various reverberatory processes as applied to the smelting of galena ores, where almost exactly the same results are produced, using lead instead of copper, and obtaining metallic lead with a minimum amount of calcination, and putting to accurate practical use the reactions just explained, although text-books on copper metallurgy are strangely silent on this important subject.

The length of time requisite to roast a charge of ore of a given weight in the long furnace under discussion depends, of course, upon the composition of the charge and the degree of thoroughness in oxidation desired. Each of the four hearths of this furnace has an effective area of about 250 square feet, and can consequently receive 4,000 pounds of ore if only 16 pounds to the square foot are charged. This is a very moderate charge, especially for heavy sulphide ores, but will ordinarily give better results than a heavier burden. It will cover the hearth about 2¼ inches deep when charged, increasing in bulk to about 4 inches at the completion of the process. By shifting each charge every four hours, the ore will remain 16 hours in the furnace, a time generally ample to produce the desired effect. On this basis, the furnace would put through 12 tons in twenty-four hours, which may be regarded as its maximum capacity on such ores as the Butte concentrates. But this is the extreme limit for two men per shift, nor will these figures be reached under ordinary circumstances. Two cords of wood or 2,240 pounds of soft coal should supply the grate for twenty-four hours, the supply of air to the ash-pit being kept at the lowest possible point. The sulphur contents of the ore furnish a much greater proportion of the heat than does the fuel on the grate.

The manipulations pertaining to the ordinary calcination of ore are too simple and generally known to be worthy of a place in a condensed treatise.

The following experiments form part of a series extending over several years. The author desires to acknowledge the assistance of Messrs. J. F. Talbot and F. Ames, and others, in the chemical portion of the work.

No. of Sample.	Copper.	Sulphur.	Dry Weight of Charge.	Weight Lost.	Copper in Roasted Ore.				Sulphur in Roasted Ore.	Hours in Furnace.	REMARKS.
					As Oxide.	As Sulphate.	As Sulphide.	Total.			
	Pr ct.	Pr ct.	Lbs.	Pr ct.							
1...	7.6	37.0	4,130	14.5	3.65	3.25	1.65	8.55	6.41	16	Heavy pyritous ore.
2...	7.6	39.0	4,130	11.3	2.27	3.10	2.80	8.17	11.30	12	Same ore.
3...	16.4	31.0	3,925	6.4	7.10	3.44	6.80	17.34	8.20	18	Purple ore with much pyrites and some zincblende.
4...	16.4	31.0	3,940	9.5	12.80	2.80	2.10	17.70	4.60	24	Same ore.
5...	38.8	24.3	3,600	6.2	29.20	4.40	3.70	37.30	......	18	Matte from cupola.
6...	62.2	22.0	3,580	3.7	54.90	3.80	6.60	64.30	......	18	Blue metal from reverberatory.
7...	74.8	21.4	3,800	2.4	61.60	5.40	7.90	74.90	......	18	White metal from reverberatory.

The loss of weight from the removal of the sulphur is partially balanced by the oxygen combining with the metallic bases, and is exceedingly variable, as may be seen by this table.

The loss in copper during calcination is very small, and almost entirely mechanical, being for the most part recoverable where proper arrangements are made for the deposition of the flue-dust. Average results from personal experience show a loss of about 1¼ per cent. of the original copper contents of the ore during calcination.

This flue-dust is usually of very much lower grade than the ore from which it results, being diluted with the dust from the fluxes, fuel, etc., and generally contains from 20 to 30 per cent. of its value in a soluble form, thus prohibiting the use of water as an aid to its condensation, unless provision is made to precipitate the dissolved metal.

Unless the ores treated are of remarkable purity, it is best to smelt the flue-dust by itself. Otherwise, the quality of the metal is likely to suffer, as the substances most injurious to it—arsenic, antimony, and tellurium—are volatile and sure to be condensed in the flues, thus being collected in a concentrated form.

CHAPTER V.

THE PREPARATION OF ORES FOR ROASTING.

THE various methods employed in roasting (calcining) copper ores fall naturally into two principal divisions, according to the mechanical condition of the ore, whether fine or coarse.

The main divisions may be again subdivided, according to the means employed for executing the operation of roasting. The following scheme makes a convenient working classification:

(*A*) Roasting ores in lump form.
  1. Heap roasting. Suited to both ore and matte.
  2. Stall roasting:
     (*a*) Open stalls. Suited only to ore.
     (*b*) Covered stalls. Suited to both ore and matte.
  3. Kiln roasting. Suited only to ore.

(*B*) Roasting ores in pulverized condition.
  1. Shaft furnaces.
  2. Stalls.
  3. Hand reverberatory calciners.
     (*a*) Open hearth.
     (*b*) Muffle.
  4. Revolving cylinders.
     (*a*) Continuous discharge.
     (*b*) Intermittent discharge.
  5. Automatic reverberatory calciners.
     (*a*) Stationary hearth.
     (*b*) Revolving hearth.

Coarse ore that comes from the mine in pieces of varying volume must be broken to a proper maximum size for the operation that it is to undergo. This size varies so greatly with local conditions that it is impossible to lay down any exact rules on the subject. The matter will be considered separately for each operation.

As we invariably have more fines than we require, or can use, for the covering of the heap in ore, or stall-roasting, it follows that economy warns us to go no further in the crushing than is absolutely essential for the success of the roasting. By crushing an ore any finer than this we lose in four different ways:

1. In the extra cost of fine crushing.
2. In dust.
3. In hampering the subsequent smelting process. (This applies only where the ore is to be smelted in blast furnaces.)
4. In the increased expense incurred in roasting the excess of fines in appropriate furnaces, or of using them half raw in the smelting furnaces.

The second and fourth of these objections have been practically canceled by the introduction of automatic calciners that operate so quietly that the loss from dust is less than in heap roasting, while the cost per ton of the operation will not exceed that of the ruder method. Even where wages are very low and heap roasting is cheap, the thoroughness and uniformity of the furnace operation will often render the latter more economical. But, as the heap or stall yields a coarse product admirably suited to blast-furnace work, and also avoids the heavy outlay for a calcining plant, it will, no doubt, long be used in remote districts, and in the early stages of certain metallurgical enterprises.

Hence, it is of importance to crush the ore intended for heap, stall, or kiln-roasting, in such a manner as to make the smallest possible proportion of fines, providing, always, that the method pursued is sufficiently economical and rapid.

Ores containing a high percentage of sulphur—25 and over—will give excellent results if so broken that the largest fragments shall be capable of passing through a ring 3 inches in diameter, and, in some cases, will roast to perfection, if sufficient time be given, in lumps the size of a man's head, while more rocky ore, which is likely to be of a harder and denser texture, should be reduced to pass a 2-inch opening. Careful experiments can alone determine the most profitable size for any given material, and should be continued on a large scale until the metallurgist in charge has fully satisfied himself on this point. This may be determined with the least trouble and expense by noticing the weight and quality of the matte obtained by smelting the roasted ore from various heaps formed of fragments differing in their maximum size.

All other conditions being identical, the heap that yields the smallest quantity of the richest matte has, of course, undergone the most perfect oxidation, and should be selected as a standard for future operations. Variations that may occur in the chemical or mechanical condition of the ore should be carefully watched as a guide in fixing upon the best roasting size. Local conditions must determine whether a jaw-crusher or hand labor should be used for this purpose. The production of fines is a decided evil in the preparation of ore for heap roasting, and the manual method possesses a certain advantage in this respect, though this consideration may be outweighed by other economic conditions. A trial of the comparative amount of fines produced by machine and hand-breaking was carried out on three different varieties of sulphureted copper ores of average hardness, and aggregating 2,220 tons.* The broken ore was thoroughly screened; all passing through a sieve of three meshes to the inch (6 mm. openings) was designated as fines.† One-half (1,110 tons) of this material was passed through a seven by ten jaw rock-breaker, with corrugated crushing-plates (which produce a decidedly less proportion of fines than the smooth plates). The breaker made 240 revolutions per minute, and had a discharge opening of 2½ inches. The other moiety was broken by experienced workmen, with proper spalling-hammers, into fragments of a similar maximum size. The result was as follows, only the fine product being weighed, the coarse being estimated by subtracting the former from the total amount:

BROKEN BY JAW CRUSHER.

	Tons.	Per cent.
Fine product—below 6 mm. in diameter	192.25	17.32
Coarse product—between 6 mm. and 64 mm.	917.75	82.68
Total	1,110.00	100.00

BROKEN BY HAND HAMMERS.

	Tons.	Per cent.
Fine product—below 6 mm. in diameter	103.34	9.31
Coarse product—between 6 mm. and 64 mm.	1,006.66	90.69
Total	1,110.00	100.00

* Unless otherwise indicated, all tons equal 2,000 pounds.

† It should also be explained that, owing to the large and very variable amount of fine material contained in the ore before crushing, as it came from the mine, it was passed over the screen just referred to before being either fed to the crusher or spalled by hand. Without this precaution, the results of the trial would have been valueless, as the variation in the amount of fines in the original ore was far greater than the discrepancy in the amount produced by the two different methods of crushing.

These results are quite in accordance with impressions derived from general observation, and, as will be noticed, prove that, with certain classes of ore, mechanical crushing produces nearly double as much fines as hand-spalling. As 10 per cent. of fines is an ample allowance to form a covering for any kind of roast-heap—and better results are obtained when the same partially oxidized material is used over and over again as a surface protection—it may frequently occur that, in spite of its greater cost, hand-spalling may prove more profitable than machine-breaking. This is a matter for individual decision, and can be determined only after a mature consideration of the difference in expense of the two operations, the means at hand for the calcination and subsequent advantageous smelting, of the increased quantity of fines, and whatever other factors may bear on the case in hand. The following steps should be carried out, whichever method is decided upon. The ore, after breaking, should be separated into three classes, the largest including all the product between the maximum size and one inch (25 mm.); the medium size, or ragging, consisting of the class between 25 mm. and the fine size (three meshes to the inch, which would give openings of about 6 mm. net); and the fines, as already explained. Roughly speaking the percentage of each class, including the fine ore that is invariably produced during the operation of mining, may be represented by the following figures:

Coarse	55	per cent.
Ragging	25	"
Fines	20	"
Total	100	"

This classication is effected with great ease and economy in case machine-breaking is decided upon, by the use of a cylindrical or conical screen of $\frac{3}{16}$-inch boiler iron, about 10 feet in length and 48 inches or more in diameter. This is placed below the breaker so that it receives all the ore. It is made to revolve from 12 to 16 times per minute, and has a maximum fall of an inch to the foot. This can easily separate 20 tons of ore per hour, and by proper arrangement of tracks or bins, discharge each class into its own bin. One fault in this very simple classifying apparatus is, that the coarse lumps of ore must necessarily traverse all the finer sizes of screen, thus greatly augmenting the wear and tear.

This objection, though frequently valid under other circumstances, has but little weight when it is remembered that even the smallest holes (6 mm.) are punched in iron of such thickness ($\frac{3}{16}$ inch) that it will withstand even the roughest usage for many months. To produce the three sizes just alluded to, the screen requires two sections, with holes respectively 6 mm. and 25 mm., of which the finer size should occupy the upper 6 feet, and the coarser the lower 4 feet of the screen. In remote districts, where freight is one of the principal items of expense, heavy iron wire cloth may be substituted for the punched boiler iron, and if properly constructed and of sufficiently heavy stock, will be found satisfactory, lasting about one-half as long as the more solid material. The difference in size between a circular hole 25 mm. in diameter and a square with sides of that length, should not be overlooked in changing from one variety of screen to the other. The mouth of the crusher should be level with the feeding-floor, and the latter should be covered with quarter-inch boiler iron, firmly attached to the planks by countersunk screws, by which arrangement the shoveling is greatly facilitated. With such a plant, three good laborers will feed the breaker at the rate of 20 tons an hour for a 10-hour shift, provided none of the rock is in such masses as to require sledging, and that the ore is dumped close to the mouth of the breaker. A nine by fifteen jaw-breaker of the best and heaviest make is capable of crushing the amount just mentioned to a maximum size of 2½ inches, provided the rock is brittle, heavy, and not inclined to clog the machine.

The expense per ton of breaking, sizing, and delivering into cars with such a plant, operating upon ores of medium tenacity, is as follows, the figures being deduced from average results of handling large quantities under the most varying conditions. It is assumed that the breaker is run by an independent engine of sufficient power, while the wages of an engineer and firemen are partially saved by taking the steam from the boilers that are supposed to supply the main works:

COST OF BREAKING ORE BY MACHINERY WITH A PLANT OF 200 TONS CAPACITY IN TEN HOURS.

		Per shift.	Per ton.
Power—per day of 10 hours, at one cent per ton ...		$2.00	$0.0100
Labor:			
Four laborers at $3.00		12.00	0.0600
Repairs:			
Toggles and jaw plates, etc.	$0.85		
Wear of tools, Babbitt for renewing bearings, etc.	0.75		
Daily slight repairs on machinery	0.80		
Miscellaneous items, sampling etc.	0.75	3.15	0.0157
Sinking Fund:			
To replace machinery at 10 per cent. on original cost		1.40	0.0070
Total ...		$18.55	$0.0928

If it should seem at first glance that 10 cents per ton is an unreasonably low figure, it will be noticed that the cost of transportation both to and from the breaker is not included in this estimate; the former is usually charged to mining expenses, and the latter to heap-roasting. Ore that is to undergo roasting in kilns for the purpose of acid manufacture must be broken considerably smaller than that just described, and this, of course, lessens the capacity of the apparatus and proportionately increases the expense. An increase of 50 per cent. in the above estimate will be sufficient to cover it. The figures given above have been frequently attained by the author, but only under certain favorable conditions, among which are: Abundance of power to run the breaker to its full speed, regardless of forced feeding. A constant sytem of supervision by which the plant is kept up to its full capacity of 20 tons per hour, and which demands exceptionally good men as feeders. A frequent inspection of the machinery, and renewal of all jaw plates, toggles, and other wearing parts, before the efficiency of the machine has begun to be impaired; all of which repairs should be foreseen and executed during the night shift or on idle days. A perfect system of checking the weight of all ore received and crushed, without which precaution a mysterious and surprisingly large deficit will be found to exist on taking stock. It is hardly necessary to mention that all bearings that cannot be reached while the machinery is in motion must be provided with ample self-oilers, and since clouds of dust are generated in this work, that unusual care must be taken in covering and protecting all boxes and parts subject to injury from this cause. Unless the ore is sufficiently damp—either naturally or by artificial sprinkling—to prevent this excessive production of dust, the feeders should be required to wear some efficient form of respirator; otherwise, they are likely to receive serious and permanent injury, the fine particles of sulphides being peculiarly irritating to the lungs and entire bronchial mucous membrane.

The breaking of ore by hand hammers, technically denominated "spalling," is worthy of more careful consideration than is generally bestowed upon it. The style of hammer is seldom suited to the purpose, though both the amount of labor accomplished and the personal comfort of the workmen depend more upon the weight and shape of this implement and its handle than on any other single factor save the quality of the ore itself. There should be several cast-steel sledges, differing in weight from 6 to 14 pounds, and intended for general use in breaking up the larger fragments of rock to a size suitable for the light spalling-hammers. Each laborer should be provided with a hammer 6 inches in length, forged from a 1½-inch octagonal bar of the best steel, and weighing about 2¾ pounds. This should be somewhat flattened and expanded at the middle third, to give ample room for a handle of sufficient size to prevent frequent breakage. The handles usually sold for this purpose are a constant source of annoyance and expense, being totally unsuited to this peculiar duty. It is better to have the handles made at the works, if it is possible to procure the proper variety of oak, ash, hickory, or, far better than all, a small tree known in New England as ironwood or hornbeam, which, when peeled and used in its green state, excels most other woods in toughness and elasticity. The handles should be perfectly straight, without crook or twist, so that, when firmly fastened in the eye of the hammer by an iron wedge, the hammer hangs exactly true. Their value and durability depend much upon the skill with which the handles are shaved down to an area less than half their maximum size, beginning at a point some six inches above the hammer-head and extending for about ten inches toward the free extremity. If properly made and of good material, they may be made so small as to appear liable to break at the first blow; but in reality they are so elastic that they act as a spring, and obviate all disagreeable effects of shock, wear longer, and do more work than the ordinary handle. Such a handle has lasted five months of constant use, in the hands of a careful workman, whereas one of the ordinary make has an average life of scarcely four days, or perhaps thirty tons of ore. Where the ore is of pretty uniform character, it is advantageous to adopt the contract system for this kind of work. A skillful laborer, under ordinary conditions, will break seven tons of rock per 10-hour shift to a size of 2½ inches,* taking coarse and fine as it comes, and in some cases he is also able to assist in screening and loading the same into cars. This latter operation should be executed with a strong potato-fork having such spaces between the tines as to retain the coarsest size, while the finer classes are left upon the ground. These forks are made for this purpose by a firm in St. Louis, and are much superior to the ordinary forks. When a sufficient quantity of the finer classes has accumulated and the pile or stall is ready to receive its outer layer of ragging, the mixed material should be thrown upon a screen inclined to an angle of about 48 degrees and having three meshes to the inch. This screen is elevated upon legs to such a height that the coarser class that fails to pass its openings will be caught in a car or barrow, while the fines fall either into a second movable receptacle or upon the floor, being in the latter case prevented from again mixing with the unscreened ore by a tight boarding on the front and sides of the screen frame. The amount of space required for convenient spalling is about 40 square feet per man, which will allow for ore-dumps, tracks, sample boxes, etc. A good light is essential, especially if any sorting is to be done, and it is in this case and where fuel is expensive that hand-spalling frequently presents especial advantages. When the ores are siliceous, a mere rejection of such pieces of barren quartz or wall rock as have accidentally got among the ore, or first become visible on breaking up the larger masses, may have a most beneficent influence on the subsequent fusion. Where the expense of treatment is high and work is conducted on a large scale, the profit resulting from raising the average contents of the ore even a single per cent. is hardly credible, even aside from the increased fusibility due to the diminished proportion of silica.

The cost of spalling an ore of the same character as that on which the foregoing estimates for machine-breaking are based, has been calculated from the average results of a very large quantity

* Unless otherwise specified, the term "day" or "shift" may be understood to signify the ordinary working day of ten hours, from seven A.M. to six P.M., with one hour for dinner.

of ore, assuming 100 tons to be spalled, screened, and loaded in ten hours.

COST OF SPALLING ORE BY HAND WITH AN OUTPUT OF 100 TONS PER TEN HOURS.

		Per 100 tons.	Per ton.
Labor : *			
14 men breaking ore, including screening and loading, at $1.50	$21.00		
4 men sledging and loading at $1.50	6.00		
1 foreman	2,50	$29.50	$0.295
Repairs :			
Including new steel and handles.			
5 handles at 30 cents	1.50		
7 pounds of steel at 15 cents	1.05		
Blacksmith's and other work on above, ⅓ day	1.00		
Screens, forks, and shovels	1.67		
General repairs	0.55	5.77	0.0577
Sinking Fund :			
To replace screens and permanent fixtures		0.15	0.0015
Total		$35.42	$0.3542

Perhaps the most marked point of difference between the roasting of lumps and fines is the time requisite for their oxidation. Oxidation is almost instantaneous for an infinitely small particle of any sulphide, and the time increases with the cubic contents of the fragment, until such a size is reached that the air fails to penetrate the thick crust of oxides formed upon the outside of the lump of ore or matte, and all action ceases.

It might seem, therefore, that the process of pulverization should be pushed to extreme limits, and that the best results would be obtained from the most finely ground ore. But this is by no means the case in actual practice; for other conditions arise that more than counteract any advantage in time. The chief of these, aside from the difficulty and expense involved in the production of such fine pulp, are the losses in metal, both mechanical and chemical, that occur with every movement of the ore, and reach an enormous aggregate before the operation is completed; and the liability to *fritting* or sticking together in the calcining-furnace, regardless of the greatest possible care in this process.

* I assume that wages are low; otherwise machine breakers would be used.

The oxidation of the particles takes place with such rapidity that a temperature is generated above the fusion-point of ordinary sulphides.

Still further objections could be mentioned; but those already adduced are sufficient to limit the degree of pulverization for the principal portion of the ore, although a greater or less proportion, according to the machinery used for the purpose, is crushed to an impalpable dust, and causes a considerable mechanical loss, in spite of all provision for its prevention.

The best size to which to crush varies with each individual ore, and is entirely a matter of trial and experience; nor should any one responsible for the calcination of any given material rest satisfied until he has determined by actual and long-continued experiment, that the substitution of either a coarser or a finer screen for the size in use will be followed by less favorable results.

This may be arrived at by careful comparative determinations of the residual sulphur contents after the calcination of material crushed through screens of various sized mesh and roasted for the same length of time, careful consideration also being given to the cost of crushing in each case, to the condition of the oxides of iron present (the sesquioxide is an unfavorable constitutent in reverberatory smelting), and, above all, to the quantity of flue-dust formed, and loss of metal by volatilization.

It is evident that such diverse and obscure questions can only be accurately determined by extensive and long-continued trials. But the result is well worth the labor, and in these days of almost universal information and close competition, it is only by such means that any decided advantage can be obtained.

While mattes, speiss, or similar products of fusion must always be granulated or pulverized to the degree required for calcination, it is not an uncommon quality of sulphide ores either to decrepitate, or else to fall to pieces when heated by the mere moving from place to place in the furnace, to such an extent that the charge may be made up of pieces from the size of a walnut down, without affecting either the time requisite for the oxidation or for its perfection. The product will be an almost homogeneous and impalpable powder.

A more striking illustration of such a condition of affairs can hardly be found than in the case of the concentrates from the Parrot Company's mine at Butte, Montana.

In this instance, the process of subdivision resulted from two different causes. The iron pyrites that forms the larger portion of the ore decrepitates into very minute cubes, which are subsequently reduced to a fine powder by oxidation, while the fragments of pure copper ore—bornite—seem gradually to diminish in size by the wearing away of the surface as it becomes earthy and friable from the superficial formation of oxides.

This latter phenomenon may also be observed to a less extent in the calcination of mattes when they are of a sufficiently soft or porous nature; but in roasting a considerable quantity of a very low-grade matte (from 10 to 15 per cent. of copper) that had been obtained in hard polished granules by tapping into water, it was found impossible materially to alter either the size or shape of the grains, many of which were as large as an army bean, or satisfactorily to reduce the percentage of sulphur, even by long exposure to a temperature closely approaching its fusion-point.

On the other hand, quite satisfactory results are obtained in the case of richer matte (from 30 to 40 per cent. of copper) by granulation in water; and, in many of the foreign works, this is the only means provided for the preparation of the matte for the process of roasting; but it must be remembered that this practice is confined to the English reverberatory method, where it is not desired to remove more than 50 per cent. of the sulphur by roasting, and where a portion of sulphides still remains in the calcined matte that would be entirely unsuited to the so-called "blast-furnace" method of matte concentration in cupolas, as usually practiced in this country.

Although the results described, as obtained by granulation, may be improved upon by careful attention to the temperature and pitch of the matte when tapped, and especially by care and experience on the part of the smelter, this practice cannot be recommended, excepting under peculiar conditions and in remote situations where improved crushing machinery is not obtainable, or where the physical condition of the matte is particularly favorable to the production of porous and friable granules. Nor is anything gained by its employment for the purpose of avoiding the preparatory breaker, and obtaining at once a material sufficiently subdivided for immediate treatment in the final pulverizing apparatus; for, although in this practice the larger granules are broken and crushed into a condition favorable for the calcining process, a large proportion of the entire mass is already so small as to pass through the crushing apparatus untouched, in the shape of minute spherical pellets or globules which present the least possible surface to oxidation, and retain a hard, glossy surface. These grains are scarcely affected by any moderate temperature, and may even undergo complete fusion without any perceptible loss of sulphur. Not many years ago, the question of economy might have influenced the adoption of this practice; but at the present time, and in view of the improved and comparatively inexpensive machinery at our disposal, it is probable that the inconvenience, danger, and other drawbacks inseparable from the projection of large quantities of molten sulphides into water, and their subsequent recovery from the reservoir or whatever vessel is employed for the purpose, more than outweigh the cost of crushing by machinery.

It is impossible to lay down fixed rules for the degree of pulverization of any material best suited to roasting. Each case must be decided according to its own peculiar conditions, including the cost of labor and power, and the capacity and quality of the mechanical means available.

Bearing in mind the results that may, in certain exceptional cases, come from decrepitation, it may be assumed that reduction in size beyond one-eighth of an inch is seldom advantageous in treating ores, and that the presence of a large proportion of sulphides, or of a particularly porous or friable gangue, may permit an increase of the screen mesh to one-fourth inch or more. With mattes, a slightly finer standard (from one-sixth to one-eighth inch) may be employed.

The proportion of the ore reduced to a minuteness neither intended nor desired, depends materially upon the means employed for crushing; and as the mechanical loss and other evils enumerated increase in direct ratio to the amount of fine dust in the charge, it is evident that, other things being equal, the apparatus best adapted to the breaking of ore or matte is that which produces the smallest proportion of fines.

CRUSHING MACHINERY.

The crushing machinery used for the purpose under discussion may be divided into two classes.

1. For preparatory crushing: Breakers of various patterns.
2. For final pulverization: Stamps, Ball pulverizers, Chili mills, various patent pulverizers and grinders, Cornish rolls.

I.—MACHINES FOR PREPARATORY CRUSHING.

Apart from machines intended for fracturing large masses of rock, such as rock-hammers, all preparatory breakers consist essentially of a movable jaw that squeezes the pieces of rock against a more or less rigid frame. The most useful types are:

The Gates crusher, and

The Comet crusher, in both of which the movable jaw is a massive cone, suspended, or supported on a step, which oscillates with a gyrating motion in a heavy, bottomless, cup-shaped mortar. This type has very great capacity.

The universally-known Blake crusher, of which there are several patterns, has a movable jaw hanging from two pivots, which is pressed against the stationary one by a pitman, or by a vertical connecting-rod and toggles.

The Blake-Challenge is a sectional machine, constructed of wrought iron and steel, the heavy thrust of the crushing being taken by two powerful steel rods.

The Blake multiple-jaw breaker is considered in the following section.

The Dodge crusher is particularly suited to reducing fragments of moderate size to still finer grains, and when built of sufficient strength to admit of the wide jaw that is necessary for large capacity, is a most useful machine. The jaw oscillates on fixed points projecting from its lower extremity, the maximum of motion being at the top of the jaw.

There are other excellent patterns of jaw-crushers that are suited to special conditions.

The principle of jaw-crushing is eminently satisfactory as regards economy, capacity, and general suitability to the purpose for which it is intended.

A machine should be selected that has stood the test of years, and is manufactured by some well-known and reputable firm. Light-built machines should be particularly avoided, as the strain exerted upon certain parts of every breaker, especially when clogged with clayey ore and set to crush fine without shortening the stroke of the jaw, is something enormous, and only to be successfully encountered by superabundant strength in every portion of the apparatus. This is well exemplified in the breakers turned out from the foundries of those manufacturers who have long made a study of this particular business, and who have gradually added an inch of metal here and a half inch there, as time and trials have developed the weak points of the machine, until it may appear bulky and clumsy beside the light and elegant models of some of their later competitors.

As the ore usually passes directly from the breaker to the rolls —better with the interpolation of a short screen to remove such as is already sufficiently fine; and,as in fine crushing the capacity of the breaker, even when set up to its closest practicable limits, usually greatly exceeds that of the rolls, a decided increase in the work performed can be most economically and easily effected by introducing a second fine breaker between the coarse crusher and the final pulverizer. This machine may be of quite light construction, should have a very long, narrow jaw opening—say 2 by 18 inches—a slight "throw," and move at a high speed.

II.—MACHINES FOR FINE CRUSHING.

The apparatus best suited for this purpose may be brought under the following heads:

Stamps.
Ball pulverizers.
Chilian mills.
Multiple-jaw crushers.
Cornish rolls.

Stamps, although universally known and always reliable, produce far too great a proportion of fine dust, besides being unnecessarily expensive, both as regards first cost and subsequent running.

The Ball pulverizer, when properly constructed, has the merit of compactness, slight cost, economy in running, and several other advantages, but is of insufficient capacity, and, like stamps, is better calculated for the production of fine pulp than of the material required for calcination.

Chilian mills are rapid, economical, and effective pulverizers, but have usually had serious faults in their construction. After much experimenting, Fraser & Chalmers have designed a mill of this pattern that is such a radical improvement on anything I have ever seen before, as to merit the attention of every metallurgist. But for calcination, the Chili mill is not well adapted, as it tends to produce a fine powder rather than the minute granules that we prefer for calcination.

Multiple-jaw Crushers.—The Blake multiple-jaw crusher embraces series of sliding jaws actuated by pitman and toggles, as in the ordinary Blake crusher. It offers the advantages of the jaw system of crushing, and can be used in crushing material as fine as can ever be required for ordinary calcination. In crushing lump ore down to a grain of 3 mm. (⅛ inch), the following set of crushers might be used:

One 7 by 10 ordinary Blake breaker.

One 3-jawed 2 by 20 multiple-jaw crusher.

One 7-jawed ½ by 24 multiple-jaw crusher.

The portion that is already crushed sufficiently fine is removed between the second and third crushers, by means of a revolving screen, the product of the last crusher also being elevated to the same screen. The extraordinary crushing surface obtained by thus multiplying the jaws is very apparent. Thus, No. 2 crusher has the equivalent of a jaw 60 inches long, and No. 3 of a jaw 168 inches long.

A saving in first cost, in power, and in dust production, are some of the most important advantages claimed for this system of crushing.

Cornish Rolls.—Few machines can compare with the Cornish roll for capacity, economy, and certainty in crushing every variety of ore and matte for the purpose just indicated. But inasmuch as the various patterns of this machine differ almost as much among themselves in efficiency and capacity as they do from the other pulverizers already mentioned, and as an examination of a large proportion of the roller plants in actual use at the present time in this country indicates a great want of care in both construction and management, and a tendency to be satisfied with a considerably lower standard of excellence than might easily be attained, it seems desirable to draw attention to such points as seem to particularly demand supervision or reformation.

Rolls should be ordered only from the best makers, who can refer to numerous similar machines of their manufacture in long and successful operation, nor should the metallurgical engineer forget that much of the work for which rolls are made, and in the performance of which they give perfectly satisfactory results, is for phosphates, gypsum, lead ore, or similar soft or brittle substances, whose crushing bears no relation to that of the low-grade matte and tough quartzose—or hard pyritic—ores that are generally the object of calcination. Certain low grades of matte, especially when produced in blast-furnaces, contain a large proportion of various indefinite compounds of copper, iron, and sulphur that are almost malleable, and would inevitably destroy any of the ordinary light-weight, low-priced rolls so frequently considered sufficient for general purposes, and occasionally placed in metallurgical establishments with mistaken notions of economy.

A volume might be written on the subject of Cornish rolls. I must confine myself to a glance at the three most important points connected with their construction and management.

(*a*) Gearing and speed.

(*b*) Springs.

(*c*) Shells, or tires.

(*a*) Geared rolls are preferred, by some engineers, for coarse crushing. For finer work, they cannot compare with rolls driven direct with belts. Geared rolls should have a speed of about 40 revolutions per minute A higher speed increases the production of dust. The fine rolls may be speeded 100 to 160 revolutions, this speed being rendered practicable by direct belting and heavy band wheels acting as fly wheels. The peripheral velocity may vary between 600 and 1,000 feet per minute.

The old system of building rolls too weak for the work they have to do, furnishing insufficient power to drive them, and then allowing them to spread apart and shirk every hard lump that happens to come between them, need scarcely be considered.*

(*b*) Springs of rubber, or preferably, steel, must be used, or a weak point must be provided, that will break when any dangerous strain is brought upon the rolls. For coarse rolls, I prefer extra strong steel springs, while breaking-cups are best adapted to fine rolls. Ordinary car-springs are not stiff enough for the coarse rolls, when running on hard, tough rock, the resistance desired under such conditions being some 50 tons or more. A good way is to group a number of such springs between two plates, and thus form a practically rigid block that bears against the movable roll in such a manner that its elasticity will not come into play until the predestined compression limit is reached. By a familiar arrangement of equalizing-levers, we distribute the strains uniformly over both bearings, that the yielding roll may not be forced into an oblique position.

Unless rolls are specially constructed for the purpose, nothing is gained in setting them so that their surfaces are in direct contact, even for the finest crushing, as they will constantly choke and give trouble, without yielding nearly as large an amount of

* I regret to say that this practice may be seen to-day in its fullest development in the new concentrator at the Elisabeth shaft of the Himmelfahrt mine at Freiberg.

product of the desired fineness as when they are set slightly apart, and the product that is not fine enough to pass the screen is returned to them.

(*c*) Shells, or tires, may be made either of chilled iron or of hammered steel. The chilled tires are so brittle, and the chilled surface frequently so unequal in quality and depth, that they often cause much annoyance. It is also a very tedious job to dress them into shape when they require it.

Hammered steel will usually prove the more satisfactory material, and can easily be turned true when worn. The life of the shells will depend largely upon the man in charge of them. By so distributing the stream of ore as to throw the maximum of work on the least worn portions of the tires, their existence can be greatly prolonged. I have crushed some 38,000 tons of hard ore with one set of such tires, and am aware that this duty has been much exceeded.

Finally, rolls, like all crushing machinery, will work with economy and satisfaction only when their capacity materially exceeds the duty that is put upon them.

The elevator is a necessary evil, but its delays and annoyances can be greatly reduced by constructing it of a capacity far beyond the requirements of the case. Chain elevators are not a success, having too many wearing parts. For heavy work I prefer to use a 12-inch six-ply rubber belt, with heavy, 10-inch steel buckets riveted to the belt. A speed of about 250 feet per minute is satisfactory. Wherever the inclination is not too great, conveyer belts running over concave idler-pulleys, are the most economical and satisfactory.

Perhaps the most useful and durable screens are steel plates punched with diagonal holes and set in a hexagonal frame. The plates can be easily renewed, and the rate of screening can be varied by changing the level of the lower bearing. Screens are seldom designed of sufficient capacity.

CHAPTER VI.

THE ROASTING OF ORES IN LUMP FORM.

I.—HEAP-ROASTING.

THE roasting of sulphureted ores or copper in mounds or heaps dates back beyond the age of history, and,in its most primitive form,is still practised among barbarous nations who have evidently never held communication with each other. It is not difficult to imagine its origin in the midst of some rude people, whose possession of superficial deposits of oxides and carbonates of copper had taught them the value of that metal as obtained by a simple process of fusion, while the sulphide ores that were doubtless encountered at a slightly greater depth were thrown aside in heaps as worthless until the spontaneous combustion of some of these waste-piles, brought about by the decomposition of the sulphides, and the interesting discovery that ores, hitherto considered valueless, would, after a simple burning, also yield the coveted metal, led some metallurgist of that day to the idea of calling in the aid of artificial combustion to hasten matters. Nor has this rude and simple process undergone that general improvement that one might have expected when considering the tremendous advances made in other appliances for accomplishing the same purposes. A somewhat careful inspection of nearly all the localities in the United States where heap-roasting is practised reveals the fact that the results obtained are far from satisfactory in the greater number of instances. The amount of fuel employed and the height and size of the heap are not correctly proportioned to the sulphur contents of the particular ore under treatment. Fragments of rock far exceeding in size the extreme proper limit, as determined by experience, are mixed with material so fine as to be fitted only for the covering layer, and these are dumped upon the ill-arranged bed of fuel without regard to the final shape of the structure or the establishment and maintenance of the requisite draught. Also, a sufficient quantity of proper material for the all-important covering layer is not applied. The result of these, and some other, deficiencies is that a small proportion only of the ore is exposed to a proper degree of heat, and the remainder of the heap is pretty equally made up of half-molten masses of clinkers from the interior, and comparatively raw and unburned material from the outer layer. With the exception of what little sulphur may have been driven off by volatilization, the ore after such a calcination is scarcely better fitted for the fusion that is to follow than if it had not been roasted. The evil results of an imperfect preliminary calcination can only be fully appreciated after the ore has passed to the next stage of treatment; in fact, they are so far-reaching that it is impossible to express the full measure of the damage in exact figures. A discussion of the effect of imperfect calcination and of its remedies will be found under the head of "Smelting Sulphide Ores in Blast Furnaces." The vital importance of the process, and the almost universal want of care and supervision in the carrying out of its details, will justify this urgent remonstrance against its improper execution. Moreover, the cost of roasting properly is no greater than that of doing it imperfectly.

The responsibility of selecting *heap-roasting* in contradistinction to the other methods enumerated for the desulphurization of an ore must rest upon the metallurgist in charge of the works, and is a question deserving the most careful consideration; nor are the reasons for or against its adoption in most cases so clear and self-evident that plain and unvarying rules can be laid down for his guidance. In this, as in many other instances, there are usually strong metallurgical, commercial, and sanitary arguments that should be carefully weighed. The contiguity of cultivated land, or even of valuable forests, would forbid the employment of heap-roasting unless the arguments for its adoption were sufficiently powerful to outweigh the annoyance of constant remonstrances on the part of the land-owners, accompanied by claims for heavy damages from the effect of the sulphurous gases. For legal reasons, as well as for various other prudential and sanitary motives, it is important to learn how this damage is effected, and to what distance its ravages may extend.

1. The damage is caused solely by sulphurous and sulphuric acids, neither arsenical nor antimonial fumes nor the thick clouds of smoke evolved from bituminous coal having any appreciable influence.

2. The most injurious effects are visible on young, growing

plants; and the more tender and succulent their nature, the more rapid and fatal are these.

3. A moist condition of the atmosphere greatly heightens the injurious effects of the gases, and as our most frequent rains occur in the spring, at the very period during which the crops and forests are in young, green leaf, more damage may be effected in a few days at this season than during the entire remainder of the year. The author has seen a passing cloud, while floating over a dozen active roast piles, absorb the sulphurous smoke as rapidly as it arose, and, after being wafted to a distance of some eight miles by a gentle breeze, fall in the shape of an acrid and blighting rain upon a field of young Indian corn, withering and curling up every green leaf in the whole tract of many acres in less than an hour.

4. As might be expected, the vegetation nearest the spot where the fumes are generated suffers the most, and the direction of the prevailing winds, in a fertile district, can be plainly determined by the sterile appearance of the tract over which they blow.

The most elaborate means for obviating this evil have been tried at the great metallurgical establishments of Europe, and vast sums have been expended in this direction. The plans pursued in England tend more toward the mechanical deposition of the offending substances in long flues and passages (the first experimenters evidently having failed to realize that the sulphurous vapors alone caused the damage) while in Germany, the more scientifically correct method of effecting condensation and absorption of the gases by means of various liquids and chemicals was pursued, but with scarcely better results. In the former case, it was soon discovered that, while the oxides of zinc, lead, arsenic, antimony, and various other substances carried over mechanically, or as gases by the draught, were condensed and deposited so completely in the canals that the air issuing from the top of the tall chimney was practically free from them, the percentage of sulphurous and sulphuric acids, which alone are responsible for damage to vegetation, was not sensibly diminished. Similar efforts in Germany for the absorption of the sulphur gases were carried out with such imperfect and ill-adapted apparatus, and on so inadequate a scale, that the absolute impossibility of a successful issue must be apparent to any one reading the pamphlet issued by the Freiberg officials intrusted by government with the execution of the experiments. But however insufficient the apparatus, the results arrived at decisively indicated the impossibility of disposing of the offending fumes by any plan of condensation or chemical absorption, except on a small scale and with ususually dilute gases.

The problem has long been solved in Europe, in the only rational and economical manner, by utilizing the hitherto destructive fumes for the manufacture of sulphuric acid. This requires, of course, the abolition of heap-roasting, and the confinement of all processes of calcination to such closed kilns and furnaces as may be placed in direct communication with the leaden acid chambers. The very secondary position held by agriculture in those sections of our country that furnish the material for the principal smelting works has, up to the preent time, obviated any necessity of dealing with this question, though some of the largest copper smelting works in the East have already adopted the European solution of the problem as a matter of profit rather than of necessity.

In the case of smelting establishments of such capacity that not more than twenty-five tons daily of sulphur are oxidized and poured into the atmosphere, it is probable that all vegetation outside of a circle of four miles in diameter may, under ordinary circumstances, be considered safe from the effects of the fumes.

No harm to man or beast has ever been authentically reported as resulting from the use as food of an article of vegetable origin that has been exposed to the corrosive influence of such gases. This is a very important point, and careful investigation and experiments have completely disproved the opposing arguments so often made against smelting works in Germany by certain stock-raisers.

In laying out the ground for roast-piles, the first point to consider is the prevailing direction of the wind, great care being taken that the fumes shall neither be blown toward the works themselves, nor toward the offices and dwelling-houses in their immediate neighborhood. Smelting-works are frequently situated in a valley, in which the prevailing winds naturally follow its longitudinal axis. In this case, a tract of ground on one side or other of the central depression, instead of in its immediate course, should be selected. By careful observation, and taking into consideration that the prevailing winds may differ at different seasons of the year, the roast heaps can generally be so placed as to give no substantial ground for claims of damage to agriculture. Care should also be taken that the selected tract is free from any possible chance of inundation; that it is either perfectly dry, or susceptible of thorough drainage; that it is not crossed by gullies or depressions that may serve as watercourses for the drainage of the surrounding hills in case of a heavy shower; that it is protected as far as possible from violent winds; that snow does not drift on it badly in winter, and that it is at least as high as the spot to which the ore is to be transported for the ensuing operation, or, if this is not feasible, at least as high as the elevator which is to raise it to the required level. If possible, it should occupy an intermediate position, as regards grade, between the shed in which the ore is prepared for roasting and the point at which the calcined product is to be delivered. A fall of 10 feet for the first step and 4½ or more for the second—total 14½ feet—will render possible the establishment of a system of handling and transportation that can hardly be excelled.

A detailed description of such a model plant will suffice as a pattern that may be varied to suit local conditions, always remembering that, under ordinary American circumstances, the economy of labor is one of the first conditions to be observed, and that the saving of 25 cents in handling a ton of crude ore is equal to a dollar or more on the ton of matte, and at least two dollars when estimated on the ton of copper.

Assuming that the metallurgist is called upon to prepare a yard for heap-roasting of ample size to contain a sufficient number of piles to furnish from 80 to 100 tons daily of calcined material, without encroaching upon the partially burned ore, and that the contour of the ground permits the requisite fall in each direction —as already explained—the following plan may be advantageously adopted:

Experience having demonstrated that an ordinary pile 40 feet long, 24 feet wide, and 6 feet high will contain about 240 tons, and burn for 70 days, to which should be added 10 days for removing and rebuilding, it follows that each pile will supply $\frac{240}{80}$ equal to 3 tons of roasted ore daily; so that 35 heaps will be needed to furnish the full amount of 100 tons daily. Allowing 36 feet for the width of each structure, and 60 feet for the length, in order to give ample room for various purposes that will be explained hereafter, an area of 75,600 square feet will be required.

The frost being out of the ground and the surface dry, a rectangular area of the extent just computed should be prepared by means of plow and scraper, being leveled to a perfect plane, and having a slight slope toward one longitudinal edge, or from a central ridge toward either side. The black surface soil should be removed, together with all sods, stumps, and remains of vegetation, and the space that it occupied replaced with broken stones, slag, or coarse tailings from the concentrator; or, best and cheapest of all, granulated slag from the blast-furnace. This can be easily obtained in any desired amount by allowing the molten scoriæ from the slag-spout to drop into a wooden trough, lined with sheet iron, placed with a grade of one inch to the foot, and provided with a stream of water running through it, equal to at least sixty gallons a minute. If sufficient fall is available, the granulated slag—graduated to any desired size by the height through which it falls, velocity and amount of water, and various other trifling factors easily ascertained by trial—is discharged directly from the launder into dump-carts, the water being drawn off by substituting a sieve of ten meshes to the linear inch for the lower eighteen inches of the wooden trough bottom. By this simple means, the best kind of filling can be prepared and delivered at the roasting-yard very cheaply, the expense of transportation hardly equaling the wages of the ordinary slag-men, who may be employed in attending to the loading of the carts and the leveling of the material when dumped. The entire area of the rectangle being raised at least two inches above the surrounding ground, a proper surface is formed by spreading upon the foundation already described a sufficient quantity of clayey loam. This should be rolled several times with a heavy roller drawn by horses, the surface being slightly dampened from time to time, until the entire area is as level and nearly as hard as a macadamized road.

Unless the climate is an unusually dry one, and the district free from snow, it will be better to use gravel instead of the loam, putting down a layer some four inches thick over the entire surface of the roast-yard. This will prevent mud, and the great loss arising from the treading of the fine ore into the same.

If the roast-yard is to be a permanency, and one is desirous of obtaining the best results with the least loss, a final covering of ore-fines should be added, the gravel being covered three or four inches deep with low-grade fines. Nor should this covering be confined merely to the portion of the ground that is to be occupied by the ore-heaps, but should be applied to the entire surface, including spaces between the heaps, passageways at ends of heaps, etc., etc. By so doing, there will always be a caked coating of ore-fines to shovel on, and the danger of getting dirt and gravel mixed with the roasted ore will be avoided completely.

As the layer of fines beneath the heaps becomes gradually roasted through, it should be removed with the coarse ore and sent to the furnaces, its place being supplied by fresh fines of the richest description, for nowhere can fine ore be roasted so free from any possibility of loss as when safely buried beneath the heap.

Nothing is more important about a roast-yard than a proper drainage system. If possible, the entire ground should slope slightly toward the lateral lower track on which the roasted ore is removed to the furnaces; and where such a gentle slope can be obtained, the drainage problem is rendered very simple and perfect; for a deep ditch run all along the upper edge of the mound, parallel with the track just referred to, will cut off all the surface water from the ground beyond, and leave to deal with only the small amount of water that falls on the roast-yard itself. This water is best removed by tile drains, laid underground, with frequent openings at suitable places, where there is no danger of fine ore being washed into the drain.

They will, of course, have their discharge through the bank-wall into the ditch that runs between the lower track and the bank-wall. Assuming a fall of some ten feet between the spalling-shed and the ground under consideration, an elevated track is constructed over the central longitudinal axis of this rectangle for the purpose of delivering the broken ore upon the heaps. Where no side-hill is available the ore is carried up on to the heaps in wheelbarrows. The trestles to support the track may consist of sets or bents of two 8-inch by 12-inch posts with 8-inch by 10-inch caps six feet long. Bents 36 feet apart and properly braced. The posts should be about six feet apart at the bottom and two or three feet apart at the top.

These bents support the trussed beams 10 inches by 12 inches, on edge, which carry the track as shown in the accompanying sketch. (See Fig. 18). These girders may be made up of 2-inch or 3-inch planks spiked together.

A fall of an inch in 12 feet will greatly facilitate the handling of the loaded car, and offer little obstruction to the return of the empty one. The track should, if possible, consist of T-rails, 12 pounds to the yard, firmly spiked to the longitudinal stringers, no sleepers being necessary; and well connected with each other by fish-plates, having two half-inch bolts at each end of each rail. All tracks throughout the entire establishment should have the same gauge; 22 inches is a convenient standard.

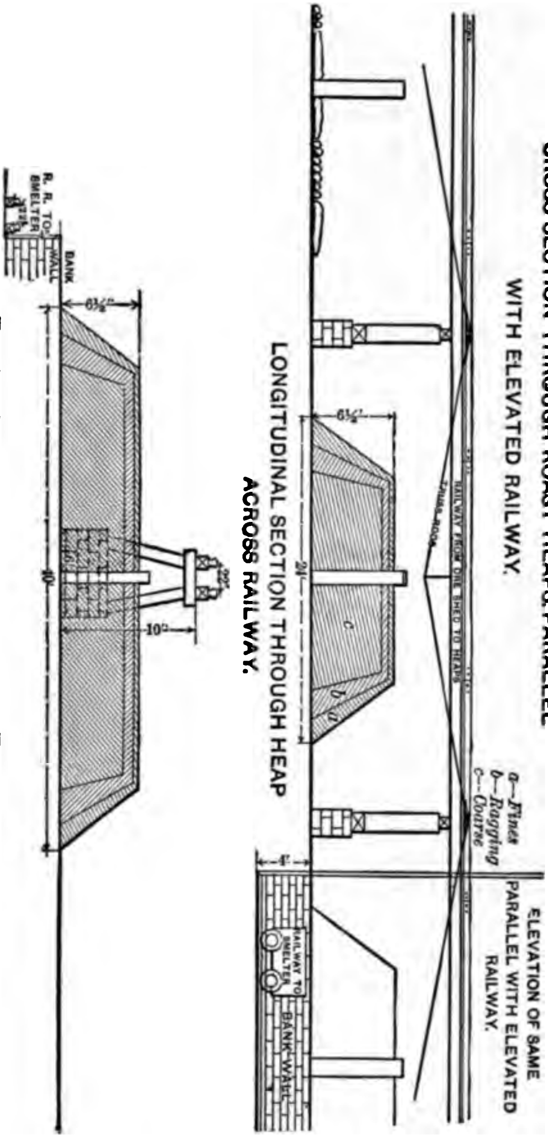


FIG. 18.—ROAST YARD WITH ELEVATED TRESTLE.

An iron-bodied end-dumping car, so made as to dump at right angles to the track, should be used. As the heaps are some 40 feet in length, the area over which the ore can be distributed by dumping from the car is far too contracted, and the following simple contrivance will be found to save many thousand dollars annually that would otherwise be expended in spreading the ore by hand; a plate of $\frac{3}{8}$-inch boiler iron, 30 inches square, fitted with a pair of short, low rails, on three sides of it, is so cut and placed upon the stationary track that the loaded car, striking first the flattened extremities of one set of the short rail pieces, while the flanges of the wheels run in corresponding slits until elevated upon the turntable by the gradually increasing height of the short rails referred to, the heavy car may be easily turned upon the greased plate by a single workman, being held and guided to the similar pair of short rails placed at right angles to those already described by a circular guard rail, fastened at that end of the plate opposite to the point of entrance. A temporary track, formed of a pair of heavy rails, held firmly together, prevented from spreading by crossties, and supported by movable trestles, is laid at right angles to the main railroad, corresponding exactly to a pair of the short side rails on the turntable plate. It will be readily seen that, by this simple contrivance, the extreme end of the longest roast-pile can be reached with the loaded car, while the turntable plate can be shifted backward and forward until every square foot of the heap has received its proper quota of ore. The accompanying dimensioned drawing illustrates sufficiently the principal arrangements described in the preceding pages. If the contour of the surface permit, one longitudinal side of the prepared yard should be bounded by a wall about four feet in height, the top of the same being level with the ground on which the roast-heaps are built, while a railroad leading to the furnaces is constructed parallel with it, in such a manner that the calcined ore may be wheeled on a plank and dumped directly into cars without having to ascend any grade, thus greatly lessening the expense of loading. The labor and cost of preparing a plant, such as has been just described, will be quickly repaid by the consequent avoidance of the waste inseparable from a moist and muddy roasting-yard, and especially from water flowing between the heaps. A case came under the author's observation, where the want of proper facilities for carrying off surface water caused a loss estimated at $12,000 within an hour, merely from the material washed away by the back-water from a swollen ditch, which passed between the roast-heaps, but which, from motives of economy, had been made too small to carry off unusual floods.

The height of the pile must depend entirely upon the character of the ore and the time for calcination at the disposal of the metallurgist. The higher the heap the more fiercely it will heat, and the longer it will take to complete the operation. Consequently, where the ore is rich in sulphur, and when time is an object, as where the supply for the furnaces is small, heaps should be made low.

An ore with 15 per cent. sulphur, which is, perhaps, as low as can be thoroughly roasted in heaps without the intermixing of a considerable quantity of fuel throughout with the rock, may be piled up to a height of 9 feet advantageously, while solid pyrites with a sulphur tenor of from 35 to 40 per cent. should never be allowed to exceed 6 or 8 feet, the measurement including only the ore, and not the layer of wood on which it rests. The best average height for ordinary ore is 7 feet, under which circumstances it will burn 75 days; the time being correspondingly diminished or increased by 10 days, if 6 inches be taken from, or added to, the above figures. The length of the heap has little influence on this time. The following table gives the result of the roasting of large quantities of various ores. In most of these cases, frequent sulphur assays were made of the ore under treatment; but in a few instances the sulphur was estimated from a general knowledge of the material. The heaps were thoroughly covered and carefully watched, and the combustion was kept at the lowest point compatible with safety, the sole object being to obtain the most thorough possible roast, regardless of time or trouble.

This should be the universal practice; for although the grade of metal to be produced in the subsequent fusion may not demand such a thorough calcination, it is better to roast a certain portion of the stock thoroughly, and then reduce, or dilute, the matte to the required standard by the addition of raw ore. This lessens expenses in various ways. It costs little or no more to roast an ore thoroughly than to do so partially; and the more completely the sulphur is eliminated from the roasted ore the larger will be the proportion of raw ore that can be used in the charge; and consequently the less will be the cost of calcining and the losses from fines of roasted ore. It is also very easy to keep the "pitch" or percentage of the matte produced at a proper point, when thoroughly oxidized stock is always at hand. These and various other reasons that could be mentioned are sufficient to refute the arguments of those who consider the addition of raw ore peculiarly injurious, and prefer an imperfect roasting.

LENGTH OF TIME CONSUMED IN BURNING HEAPS OF VARIOUS HEIGHTS.

Height in feet.	Quality of Ore.	Per cent. Sulphur.	Per cent. Copper.	Days Burning.	No. of Sample.
5	Pyrite	39	6½	54	No. 1
5	Chalcopyrite, with little pyrite in quartz	18	14.3	41	" 2
5	Bornite and pyrite	31	21.4	53	" 3
5½	Same as No. 1	39	6½	66	" 4
5½	" No. 2	18	14.3	50	" 5
5½	" No. 3	31	21.4	65	" 6
6	" No. 1	39	6½	72	" 7
6	" No. 2	18*	14.3	61	" 8
6	" No. 3	31	21.4	74	" 9
7	" No. 1, much matted	39*	6½	94	" 10
7	" No. 3	31	21.4	86	" 11
7½	Copper glance and pyrite in quartz	20*	23.4	54	" 12

The area of the heap is determined by the position and size of the ground at disposal, and the convenience of delivering the ore. Its width is limited by the distance to which the covering material can be conveniently thrown with a shovel, and by the room between the bents that support the track; 24 feet in width by 40 in length is a very convenient size, smaller heaps demanding considerably more labor and fuel to the ton of ore. With 36 feet between the bents, an ample border of 6 feet will be left on each side of the pile for collecting the fines, wheeling the same wherever required, and fully securing the wood-work against all danger of fire. Risk from fire is further obviated by elevating the foundation sill from which the uprights arise, upon a wall of slag-brick, 3 feet or more in height. A pile of the dimensions referred to, 24 feet by 40 feet square, and 6 feet high, will contain about 240 tons of ordinary ore, and should be built in the following manner:†

The corners of the rectangular space on which it is to be erected should be indicated by stakes, or, if the same size is to be perma-

*Estimated.

† If the furnaces are not too much pressed for ore it is more economical to still further increase the size of the heaps; 40 by 80 feet, and 7 feet high is none too large.

nently retained, by large stones, or better, blocks of slag, imbedded in the ground. The sides of the area being indicated by lines drawn on the ground to guide the workman, the entire space should be covered evenly to the depth of four or six inches with fine ore from the spalling-shed. This layer of sulphides answers several purposes; in the first place, it prevents the baking and adhering to the ground of the coarser ore, which, especially when much matte is formed, sticks to the clayey soil to such an extent as to tear up and injure the foundation, besides mixing worthless dirt with the ore, and causing a loss of the latter when attempts at separation are made. It also forms a distinct boundary line between the worthless and valuable materials, and, when left undisturbed during two or three operations, becomes itself so thoroughly desulphurized that the upper half or more may be scraped up with shovels and added to the roasted ore, its place being filled by a fresh supply of fines. This operation completed, the fuel is next arranged by an experienced workman in a regular and systematic manner. The quality and size of the wood is a matter of some moment, and must be determined for each individual case, it being evident that that variety of fuel that yields the greatest amount of heat for the longest time possesses the highest money value, provided the ore is of such a nature as to bear the temperature produced without fusing. As most sulphide ores will not stand the heat generated by a thick bed of sound, dry, hard wood, it frequently happens that a cheaper variety answers the purpose better. The outside border of wood that corresponds to the edges of the heap should be of better quality, as no such degree of heat is attainable there as in the interior of the pile. Therefore a large proportion of the bed may be made up of old rails, logs, gnarled and knotted trunks that have defied wedge and beetle, and such sticks of cordwood as are daily thrown out from wood-burning boilers and calcining-furnaces as too crooked and misshapen to enter a contracted fireplace. Such miscellaneous fuel causes somewhat greater labor in arrangement; but whatever the material, it must be placed with such care and skill as to form a solid and sufficient bed, varying in depth from 4 to 10 inches according to the behavior of the ore. However rough and irregular the greater portion of the fuel at our disposal may be, enough cordwood of even length and diameter should be selected to form a four-foot border around the entire heap and just within the side-lines of the area; for the even and regular kindling of the heap depends considerably upon the proper arrangement of this border. Sticks of cordwood not larger than 5 inches in diameter should be laid side by side across both ends and sides of the area. Across this layer, small wood is again piled until this four-foot border has been built up to the height of some 10 inches, brushwood and chips being scattered over the surface to fill up all interstices, while canals 6 inches wide, filled with kindlings, are formed at intervals of 8 or 10 feet, leading from the outer air and communicating with the chimneys in the center line of the heap. The empty area within this encircling border is now filled with the poorer quality of fuel, all sticks laid parallel and with as much regularity as possible, to cover all cracks and interstices, that no ore may fall through the wood, and to cover over the draught-canals in such a manner that they shall be neither choked nor destroyed by the superincumbent load.*

The chimneys, which assist materially in rapidly and certainly kindling the entire heap, are formed of four worthless boards nailed lightly together in such a manner that two of the opposite sides stand some eight inches from the ground, thus leaving spaces that communicate with the draught-canals referred to, and toward which several of the latter converse. For a heap 40 feet in length, three such chimneys, eight inches square, will suffice. They should project at least two feet above the proposed upper surface of the structure, that no fragments of ore may accidentally enter the flue opening and destroy its draught. In certain localities, where even old boards are too valuable to be needlessly sacrificed, two or three medium-sized sticks of cordwood may be wired together to form the chimney; or old pieces of sheet-iron, such as condemned jig-screens, worn-out corrugated roofing-iron, etc., may be so bent and wired as to form a permanent and sufficient passage, while this material will answer for several operations. The chimneys being placed in position, equidistant, and on the longitudinal center line of the bed of fuel, and held upright by temporary wooden supports, the heap is ready to receive the ore. This is brought in carloads of 1,500 or 2,000 pounds from the spalling-shed, and weighed *en route* on track-scales. It is dumped on a portable wooden platform about eight feet square, to prevent the deranging of the wood from the fall of so heavy a mass of rock

* An excellent paper on heap roasting in Vermont, by Mr. William Glenn, may be found in the *Engineering and Mining Journal* for December 8, 1883.

from a height of ten feet or thereabout. The first few carloads are heaped about the chimneys, and the platform is changed from place to place as convenience demands, until the bed of wood is thoroughly protected by a thick layer of ore. The remainder of the process is a very simple operation. The cars of ore are dumped in turn over the entire area by a systematic shifting of the temporary pair of rails already described, and the heap formed into a shapely pyramid, with sharp corners and an angle of inclination of some 42 degrees, or as steep as the ore will naturally lie without rolling. The main body of the structure is formed of the coarsest class of ore; the ragging is next placed upon the pile, forming a comparatively thick covering at the part nearest the ground, and gradually thinning out toward the top and on the upper surface. Its thickness depends on the amount available, and no fears need be entertained of its having an unfavorable influence on the calcination; for, when carefully separated from the finest class, a heap formed entirely of ragging will give reasonably good results. The extreme outside edge of the ore, when all is in place, should not entirely cover the external border of wood. At least a foot of uncovered fuel should project beyond the layer of ragging, both to prevent the ore from sliding off its bed as well as to insure a thorough kindling of the outer covering of mineral. The amount of wood required properly to burn a heap of 240 tons of ore will vary greatly with the composition of the latter, standing in direct proportion to its sulphur contents, and especially to the amount of bisulphides present, but may, on the average, be estimated at 6 cords, or one cord of wood to 40 tons of ore. In smaller heaps, this proportion must be considerably increased.

It is the common practice to use far too much wood in heap-roasting. This causes too great a heat at the commencement of the operation, and brings about various irregularities, such as local sintering and matting of the ore, with stoppage of air-circulation at these points, so that when the heap is finished, we find at various points several square feet of fused sulphides on the bottom. Above this comes a siliceous skeleton of an extent corresponding to the amount of matte which has been liquated out of it. And above this still, a large body of unroasted ore, which has entirely escaped the fire.

I have never seen heap-roasting more perfectly executed than at the Spanish mines of Rio Tinto. The material treated is the rich ore that is culled from the ordinary low-grade pyrites, and is re served for smelting in blast-furnaces. It carries some 8 or 9 per cent. of copper, and is a solid mass of iron and copper pyrites.

The most interesting features of this heap-roasting are the very small proportion of wood used, and the unusual height of the pile. It is built very much in the shape of a circular haystack, being some fourteen feet high in the center of the cone, and about thirty-three feet in diameter. It usually contains something over 400 tons of ore, much of it, near the center, being in pieces the size of a child's head. Only two-thirds of a cord of wood is employed, and this is distributed among twelve fireplaces, constructed roughly of rock, and spaced equidistant about the circumference of the pile. They penetrate to a depth of only four feet, so that the major portion of the pile has no wood under it. When the heap is lighted, only the small fraction of ore close to the fireplaces is kindled; and even here the amount of wood is so small that the heat is very slight and evanescent. From these twelve points at the circumference, the fire gradually creeps toward the center, while the heap is thoroughly covered with fines, and the temperature kept far lower than in ordinary heap-roasting. It consequently takes six to nine months to burn one of these 400-ton piles. But the result is a triumph of skill, scarcely ever a pound of matte being formed, while in the various heaps which were completed and partially demolished for the cupolas, I was unable to find even a fragment of unoxidized, or badly roasted ore. With the exception of a few kernels, the lumps were oxidized to their very center. I was informed that any increase in the amount of wood used to kindle the pile was a drawback rather than an advantage.

I have no doubt that we use far more wood than is conducive to good roasting, while to attempt to hurry a process, at the expense of doing it properly, is certainly not profitable. Conditions differ too greatly to admit of any hard and fast rules in such matters, and every metallurgist must determine for himself just how perfectly it will pay him to carry out this process, and how long a time he can afford to spend in doing it. He will probably arrive at the conclusion, if he gives the subject proper attention, that it will remunerate him handsomely to roast far more slowly and more perfectly and with less wood than he has ever attempted to do before. For the ore that is tied up in roast-heaps can only be charged with what it has cost—that is, the expenses of mining, crushing, and putting into heaps—and the interest, at 10 per cent. on the cost of even 50,000 tons of ore at $3 per ton, is only $41 per day; a sum completely insignificant compared with the gain arising from the better grade of matte and the lessened troubles in furnace management that will result from even a very slight average improvement in the roasting process.

The fine ore that is to form the external layer, and on which depends largely the success of the process, is seldom placed upon the top of the heap until after it is fired. Perhaps the most judicious practice is to cover the sides of the pile with a very thin layer, scattering it evenly with a shovel, and leaving the upper surface, as well as a space eighteen inches broad at the bottom uncovered; for if the fine ore is thrown carelessly upon the lower circumference of the pile, the draught is decidedly hampered and the fire stifled before getting fairly under way. For an average ore, an amount of fines equal to 10 per cent. of its total weight is ample; of this, eight tons may be strewn lightly upon the sides of the heap as just described, the remaining 16 tons—assuming the entire contents to be 240 tons—being arranged in small piles upon the empty space between the roast-heaps, where it is easily accessible to the shovel. The lighting should be done just as the day shift is quitting work, as the dense fumes of wood smoke, strongly saturated with pyroligneous acid and the various gaseous compounds of sulphur and arsenic, among which sulphureted hydrogen is always plainly distinguishable, are almost unbearable.

If possible, fine weather should be selected for this purpose; for although no ordinary rain is capable of extinguishing a well-lighted roast-heap, it may still interfere greatly with kindling a new one, and is quite likely to cause subsequent irregularities in the course of the process. There are several different methods of firing a roast-heap—such as lighting it only on the leeward side, and letting the fire creep back against the wind, kindling it through the draught-chimneys, etc., each of which has its advocates among roasting foremen; but long-continued observation has shown that no advantage is gained by any of these irregular methods, and the most sensible and successful practice is to light it as quickly and thoroughly as possible by applying a handful of cotton waste, saturated with coal oil, or a ladle of molten slag, to the kindling-wood at the mouth of each of the draught-canals, these being some ten or more in number, as already described. As the success of the entire operation depends principally on the management of the heap for the first few days after kindling, it will be necessary to study somewhat in detail the phenomena that it should normally exhibit during this critical period, always bearing in mind the impossibility of laying down any fixed rules that shall apply to all circumstances and to every variety of material.

Under ordinary circumstances, the heap may best be left entirely to itself for from four to six hours after lighting, care merely being taken that the kindling burns freely, and that the draught-holes communicate with their respective chimneys. At the expiration of this time, if the fire has spread well over the entire area, about one-half of the remaining fines that have been provided for covering should be scattered lightly upon the heap; the lower border and upper surface, which have hitherto been left unprotected, now receive a thin application, while the lateral coating is rendered somewhat thicker and more impervious. If matters pursue a normal course, the early morning—twelve hours after firing—should see the heap smoking strongly and equally from innumerable interstices produced by the settling of the whole mass, due to the disappearance of the thick foundation of fuel. Dense pillars of opaque, yellow smoke, smelling strongly of sulphurous acid, arise from the site of each chimney; although if these were constructed of wood, no sign of them will remain except a few charred fragments, resting in a slight depression, which marks their sites. The entire surface will be found damp and sticky, and the covering material will have already formed quite a perceptible crust, from the adhesion of its particles. This "sweating," as it is termed, arises from the distillation products of the fuel—owing to its very imperfect combustion—and from the moisture contained in the ore. A yellowish crust surrounding the vents from which the strongest currents of gas are seen to issue indicates the presence of metallic sulphur, the volatilization of the first loosely bound atom of which begins soon after the wood is fairly lighted. Its quantity depends on the proportion of bisulphides in the roast, as well as on the freedom with which air is admitted; the scarcity of oxygen and a high temperature favoring its direct volatilization, while an abundance of air and a moderate heat influence the plentiful generation of sulphurous acid.

During this first day, the newly kindled heap will require close and constant attention to prevent any undue local heating; nor is it at all uncommon to find that some neglected fissure has increased the draught to such an extent as to cause the sintering or partial fusion of several tons of ore at that point. The principal signs by which the experienced eye judges of the condition of affairs are the color of the gas and the rapidity with which it ascends; the amount of settling and consequent fissuring of the covering layer; and, above all, the degree of heat at different parts of the surface. A light, bluish gas, nearly transparent, and ascending in a rapid current, is a sign that the heat is too great at that point, and the admission of air too free. The fissuring of the crusted covering material, after the general and extensive sinking caused by the consumption of the fuel, indicates a rapid settling that can only arise from the melting together, and consequent contraction, of the lumps of ore. All these conditions are met by a single remedy; that is, covering the surface at that point more thoroughly with fines, by which means the air is excluded, the rapidity of the oxidation process diminished, and the temperature lowered. It should not be supposed that, because the interstices that exist in the upper part of the heap alone show evidences of heat and gas, those cracks and openings that have been left nearer the ground are of no importance; these are the draught-holes, while the former constitute the chimneys, and it is to the condition of the lower border of the pile that our attention should be most frequently directed in regulating the proper admission of air. A few shovelfuls of fine ore judiciously applied at the base of the heap will often have more effect than a carload, scattered aimlessly over the surface.

Only an experienced laborer can manage a roast-heap to the best advantage, nor is it possible to establish fixed rules for the guidance of this process, varying conditions demanding totally different treatment. In a general way, it may be said that, after somewhat subduing the intense heat caused by the sudden combustion of so large an amount of wood, the attendant should confine himself to scattering material in a thin layer over the sides and top of the structure, and effectually stopping up such holes and crevices as seem to be the vents for some unusually heated spot below.

By the third day large quantities of sublimated sulphur will be found upon the surface, in many places melting and burning with a blue flame. It is now necessary for the attendant to ascend to the top of the heap, to properly examine the upper surface, and place additional covering material on such portions as still seem too hot. In doing this, a disagreeable obstacle is encountered in the clouds of sulphurous gas, which, to one unaccustomed to the task, seem absolutely stifling. By taking advantage of their momentary dispersion by currents of air, and retreating when they become too thick, no difficulty need be experienced in covering the upper surface of the heap as thoroughly and carefully as any other part of it.

If the process of combustion seems to have spread equally to all parts of the pile, nothing need now be done except daily to scatter a few shovelfuls of fines over such heated spots as seem to require it; but if any isolated corner of the heap has failed to kindle, or, having once caught fire, has now become cold and ceased to smoke, it is necessary to draw the fire in that direction. This can be accomplished with ease and certainty by any one accustomed to the work; for there is no danger of a roast-heap becoming extinguished when once fairly kindled. Certain isolated spots—especially corners and angles—may fail to become properly ignited, but by opening a few draught-holes in the neighborhood the fire will surely spread wherever unburned sulphides still exist. Beginning at the end of the first week, and continuing for a month or more, a certain amount of sulphur may be obtained by forming 18 or 20 circular, ladle-shaped holes about 14 inches in diameter and 7 inches deep in the upper surface of the heap, and lining them carefully with partially roasted fine ore, so that they may retain the molten metalloid. The impure sulphur may be ladled out twice a day into wooden molds; but the impurity of the product, caused by the great quantity of ore-dust and cinders constantly falling into the melted material, and the extremely scant production of a substance that is hardly worth saving, discourages the general adoption of the practice, although at some of the older German works it is still kept up. Experiments made with the greatest possible care saved only one-tenth of one per cent. of the total weight of the ore from a 30 per cent. bisulphide ore.

With certain varieties of ore, the sulphur, instead of collecting in a concentrated form at the principal issuing vents of the strongest currents of gases, condenses over the entire surface in a thin layer, and upon melting,cements and agglutinates the fine particles of the covering layer in such a manner as to form an almost impermeable envelope. In such cases this crust must be destroyed, from time to time, with an iron garden-rake, or the process of calcination may be delayed for weeks beyond its customary limit from the lack of sufficient oxygen to maintain the proper rate of combustion. If arsenic is present, even in the smallest quantities, it will soon make itself visible as beautiful orange-colored realgar, AsS, and minute clusters of white, glistening crystals of arsenious oxide, which usually form at the upper orifices of the accidental draught-canals that communicate with the interior of the heap.

A strong and persistent wind from any one direction has an unfavorable effect on the process of heap-roasting, driving the fire toward the leeward side, and cooling those portions that feel the direct influence of the air-current to such an extent that one-fourth or more of the heap may remain in a raw condition. It is a somewhat remarkable fact that, while it is almost impossible to quench a roast-heap with water, unless completely flooded for a considerable length of time, a simple excess of the very element most favorable to its perfect combustion should have the power to extinguish it. If this annoying circumstance repeats itself with any frequency, it will be necessary to erect a high board fence on that side of the yard whence the most persistent winds prevail. Moderate rain and snow have little influence on the course of the process, except in so far as they may cause serious chemical and mechanical losses. It is only after a heavy shower or sudden thaw that the great advantage of numerous and well-preserved ditches surrounding the entire area, and even leading between the heaps themselves, is fully realized and appreciated. When wet weather supervenes, after a long period of drought, the amount of copper dissolved from the soluble sulphate salts formed during the extended term of dryness may be so large as to repay some efforts to recover it. By simply leading the drainage from the roast-yard into two old brewer's vats partially filled with scrap-iron, during one summer, 35,546 pounds of 40 per cent. precipitate were collected.

We have already pointed out the necessity of guarding against this loss by every possible means at our disposal; but even with every care a considerable loss from this source cannot be avoided in any ordinary climate.

Mr. Wendt* gives some important figures bearing on this point, relating to heap-roasting as formerly practised at Ducktown, Tenn., where, however, the rainfall is exceptionally great. We quote also his estimates of cost, which, taking into account the low cost of fuel and labor, correspond closely with our own.

"Ore-roasting, as thus carried out (in heaps), was a very economical process in point of labor and fuel. On an average, one cord of wood was consumed for 40 net tons of ore for each fire. The cost of labor in the first fire was 5 cents per 1,000 pounds for

* See *The Pyrites Deposits of the Alleghanies*, by A. F. Wendt, New York, 1866, page 19.

both Mary and East Tennessee ores; for the second fire, 7 cents and 6 cents respectively were paid; and for fine ores, the pay was 12 cents per M.

"The exact cost per net ton of ore was as follows:

$\frac{1}{20}$ cord of wood at $3	$0.15
Labor, 1st fire	.10
Labor, 2d fire	.14
Materials	.03
Total, per ton	$0.42

"The losses of copper in the above-described roasting have been very generally ignored in judging of its expense. At least, proper emphasis has never been laid on them.

"Owing to an unexplained difference of several hundred thousand pounds between the fine copper produced at the Ducktown smelter during a period extending over several years, and the monthly fine copper statements arrived at by deducting one and one-quarter unit from the assay value of the ores produced, the writer's attention was forcibly called to this subject. A careful series of experiments was instituted; the results were rather startling. Repeated analysis of ore weighed into a roast-pile, and analysis and weighing of this same ore when sent to the matte furnaces, proved an almost incredible loss.

"From the large number of experiments and analyses, I quote the following striking examples:

PILE No. 349.—MARY ORE.

Gross Weight of Ore.	Per Cent. Water.	Per Cent. Copper.	Fine Copper, Pounds.
399,213	2.5	5.0	19,461
204,444	2.0	5.8	11,620
95,182	3.8	5.0	3,617
8,663	3.0	5.1	428
34,165	6.0	4.0	1,284

741,667 pounds raw ore contained 36,410 pounds copper.

"The pile after roasting weighed 741,716 pounds—assayed 3.31 per cent. copper—equivalent to 24,985 pounds fine copper; 11,125 pounds copper, or 31.4 per cent. of the contents of the pile, had been lost while roasting; 170 days were consumed in roasting the ore and 69 days in removing it to the smelting-furnaces. Hence, the ore lay exposed to the weather for 239 days, that is, eight months.

PILE NO. 447.—MARY ORE.

Gross Weight of Ore.	Per Cent. Water.	Per Cent. Copper.	Fine Copper, Pounds.
172,882	3.0	4.7	7,881
1,532	5.5	6.3	91
198,800	2.0	4.5	8,767
32,178	4.0	5.3	1,637
26,865	5.5	4.6	1,167
32,245	3.0	6.2	1,939

464,505 gross pounds ore contained 21,482 pounds copper.

"Weight of the roasted ore was 495,566 pounds, assaying 2.85 per cent., or 14,152 pounds fine copper. During an exposure of 186 days the ore had lost 34.3 per cent. of its copper.

"All the experiments made on a total of nearly 3,000 tons of ore proved, beyond possibility of doubt, an average loss of more than one unit of copper, or over 20 pounds of ingot per ton of ore. This great loss during the roasting readily accounted for the deficit in the copper production, if only 1¼ per cent. was deducted from the assay value of the ores for losses by treatment. The actual loss by the smelting process, as practised at Ducktown, approached two units. Further experiments were made to confirm the results obtained. Experiments in roasting in furnaces proved that no copper escaped in the fumes. This, indeed, was anticipated, as the heat in roasting never could reach a point at which copper is volatile. The only other possible loss is by the leaching of the roast-piles during the heavy rains frequent in the Ducktown hills; and to this cause the great losses were finally ascribed. In referring to experiments in the leaching of these ores later on, this subject will be discussed in detail. Suffice it here to say, that with a roasting in one fire only, from 1 to 1½ units of copper became soluble in water. The results were further confirmed by copper found in large quantity in the clay 'bottoms' of the roast-piles. After a shower of rain, the roast-yard would be covered with pools of green water highly charged with copper."

During the last two-thirds of the life of the roast-heap it hardly requires an hour's labor, and if the works possess an ample stock of roasted ore in advance, nothing further need be done to the pile until it has burned itself out and becomes sufficiently cool to handle. The daily inspection, however, should never be omitted; for, even at this advanced stage of the process, irregular settling or swelling of some portion of the structure may cause sufficient fissuring and consequent admission of air to produce serious matting, a disaster that the application of a single shovelful of fines at the beginning of the trouble would have prevented. In fact, it is far better to leave the heap undisturbed, unless good reasons exist for breaking into it, as the agglutinated covering material forms a roof almost impermeable to rain and wind, while the freshly calcined ore, when exposed to these elements, necessarily undergoes a serious waste. But if, as is in most instances the case, the demand for ore from the smelting department exceeds the supply from the mine, but scant time can be afforded to the intermediate steps, and the calcination must suffer. If, therefore, it is the object to utilize, at the earliest possible moment, the ore that is stored up in the heaps, they should be closely watched, and whatever portions of the same—usually the ends and corners—are found to be moderately cool, should be carefully stripped and broken into, the object being to cool the ore that is already roasted, and extinguish the last remains of fire as rapidly as possible, without interfering too seriously with the process of oxidation that is continuing in the main body of the pile. This is accomplished by digging away the calcined ore, and following up the line of fire as it recedes from the surface toward the center, without approaching it so closely as to completely extinguish it in that portion of the ore not yet properly calcined, which is easily done at this stage of the operation. At least 12 inches should be left between the outer air and the line of active oxidation, and it is a good practical rule never to allow the surface to become so hot as to be unbearable to the naked hand.

The too common practice of keeping the smelting department so far in advance of the ore supply as to require the breaking into and utilization of roast-heaps in which the ore is still red-hot, and just at the most active and profitable stage of calcination, necessitates the employment of a strong body of laborers to bring water and constantly drench the smoking ore, in order to make it at all possible for the other workmen to shovel it into their barrows, and must be condemned as unnecessary and productive of more trouble and expense than almost any other practice at our smelting works.

Among these sources of extra expense are the doubled cost of taking down and transporting the roasted material; the burning and rapid destruction of tools and cars; the medical bills claimed by the workmen who suffer from such unhealthy employment; and, far greater than all, the injurious effect on all subsequent steps of the process, which will be referred to in the chapter on "Smelting in Blast-Furnaces."

On the other hand, the only possible advantage that can be claimed is, that some two or three weeks' interest on the value of the ore is saved.

When the heap is properly cooled, the mass of ore, which, while still hot, is often almost as hard and tough as a wall of solid rock, crumbles to pieces with a single blow of the pick, and is wheeled in barrows from the roast-heap to the furnace car.

When the heap is sufficiently cooled, it is "stripped" by removing not only the fines that formed its cover, but its entire surface, to such a depth as is necessary to include all material that has escaped oxidation. This unroasted material is made up largely of the fines forming the cover, and which, though often quite thoroughly oxidized on the top of the pile, are so agglutinated with sulphur as to be unfit for the furnace. The covering of the sides is seldom sufficiently roasted, and this is especially the case near the ground, where the ragging itself, to a depth of several inches, is frequently found unscathed. The angles of the pile are also seldom in good condition, and many isolated patches and bunches of ore will be found that the careful foreman will reject. This statement, however, refers rather to the results of the ordinary practice than to those that can easily be obtained by close attention to details and by enlisting the interest of some intelligent foreman. As already explained, the fire will find its way to every nook and corner where sulphides still exist, if only the conditions are favorable. The author recollects with satisfaction the mortification displayed by his roasting foreman but a few years ago, at the unusual occurrence of a few hundred-weight of fused, and a still smaller amount of raw, ore in a heap of some 200 tons.

A half-fused, honeycombed condition of the upper part of the heap, presenting the appearance of a skeleton of gangue from which all mineral has been melted out, is a certain indication of a proportional amount of matte below. This molten material naturally gravitates to the bottom of the heap, and is there found in masses of greater or less extent; often of many tons' weight, though, in such a case, warning would have been given during the roasting by the irregular sinking of the heap, and even by depressions and crater-like cavities on the surface. This molten product is very properly termed "heap-matte," and varies but little in appearance or composition from the similar product of a blast-furnace. A popular impression prevails among certain foremen, and even assayers, that the light honeycombed material that remains after the melting out of its sulphide constituents is rich in copper, but the contrary is true. The unfused skeleton merely represents the siliceous slag, while the molten sulphide mass below is the equivalent of the matte, the purity and value of either product depending on the temperature to which the ore has been subjected, and the consequent perfection of the smelting or liquation process. This fact is sustained by the following assays of samples of considerable size:

	No. 1.	No. 2.
Original ore before roasting	21.6 copper.	18.6 copper.
Siliceous skeleton	7.3 "	6.4 "
Heap-matte	34.7 "	36.6 "

The formation of this heap-matte in any considerable quantity is very detrimental to the roasting process, but is easily avoidable; for it is invariably caused by either too much or too little air. In too many instances, no particular notice is taken of its occurrence, and it is sent to the smelting-furnace mixed with the well-roasted ore. This is exceedingly bad practice, and should on no account be permitted, as it is totally impossible to foresee the grade of matte that will be produced by the smelting process when this unroasted sulphide is mixed in unknown and varying quantities with the properly prepared charge. If the percentage of the furnace mixture be such that the addition of this raw matte does not lower the tenor of the product below the desired standard, it may then, of course, be fed with the roasted ore, but should be kept strictly by itself, and added to each charge in weighed quantities. Any infringement of this rule gives rise to the formation of a matte varying greatly in its percentage of copper as well as in its entire composition, and deranges not only the smelting process, but seriously affects the regularity of the matte concentration operations.

The heap-matte may occur in such masses that serious difficulty is experienced in breaking it up, especially as it retains its heat for a great length of time, and in this condition is almost malleable, yielding and flattening under the blows of the sledge like a block of wrought-iron. Much expense and annoyance may be spared by stripping the central molten mass thoroughly of all adhering ore, and allowing it to cool for two or three days; at the expiration of which time it will be found quite brittle and comparatively easy to deal with. Thorough and repeated drenchings with water will produce even better results; but it should be borne in mind that a considerable proportion of the cupriferous contents of calcined ore is in a soluble condition.

When through carelessness or inexperience heap-matte is formed, it must be either treated together with the matte produced from the first fusion in the blast-furnace, or set aside until a sufficient amount is collected to form a small heap by itself, and be re-roasted. It should, on no account, be mixed with the raw ore, as it demands a different treatment, and will either cause irregularities in the ore-roasting, or will pass through that process unaltered and with no perceptible diminution in its percentage of sulphur.

The proportion of strippings and other unfinished products of heap-roasting that may be considered allowable was determined experimentally by simply weighing the finished and unfinished portions of half a dozen consecutive roast-heaps, averaging about 240 tons each. About 10 per cent. of fines were used for the covering layer in each case. The total amount of unroasted material, as given in the following table, shows that even a portion of the fines is thoroughly oxidized:

	Unroasted. Per Cent.	Roasted. Per Cent.	Days Heap was Active.
No. 1	9.6	90.4	64
" 2	6.6	93.4	71
" 3	8.4	91.6	70
" 4	9.0	91.0	61
" 5	7.6	92.4	67
" 6	11.4	88.6	57

The figures have been slightly corrected, without altering their relative values, to make the aggregate in each case exactly equal 100 per cent., which, of course, can never be precisely attained by adding the weights as actually arrived at.

While these results are taken from ordinary everyday work, it should be understood that they can only be attained by the most careful attention in the roasting-yard. The proportion of the product rejected as unfit for the smelting-furnace at some works might be even less than in the case just cited, and the reason may be readily recognized in the low grade of the product from the fusion, and the constant complaints of the impossibility of keeping the matte up to the proper standard. A selection in such cases as rigid and thorough as in those just tabulated would result in the rejection of from 25 to 60 per cent. of the entire heap.

An allowance of 10 per cent. may therefore be considered reasonable —although demanding more than ordinary care and skill—and of this, three-fourths should be fines. The stripping should be performed in a cleanly and systematic manner, and to an extent several feet in advance of the line of excavation, and the material thus removed piled on one side, to be subsequently screened on the first calm day; for the least wind causes a heavy loss when handling this half-oxidized powder. The fine part is again used as a covering, for which it is much better suited than raw ore, while the much smaller coarse portion is added to the nearest heap in process of erection.

It will be readily seen that very much more fine ore is produced during the processes of mining and crushing than can be used for the purpose of covering material, especially as only a small proportion of the latter is sufficiently oxidized at each operation to be passed on to the smelting-furnace. The problem of the best means of utilizing this constantly increasing amount of fine ore in works unprovided with calcining-furnaces is often a pressing one. It will be referred to again, under the heading, "The Treatment of Pulverized Ores."

The roast-heap, when once tolerably cool, is torn down and loaded into the furnace-car with great celerity. Three or four men trundle the barrows, while double that number wield the pick, shovel, and hammer. It is the duty of these laborers to break all partially fused masses, or lumps that are too large for proper smelting, into fragments of a reasonable size, as especially determined by the metallurgist. There is not time, or space, or opportunity on the charging-floor of a blast-furnace in full operation to attend to any duties beyond those immediately connected with weighing the charge and filling the furnace, and many serious irregularities in the smelting may be traced to an omission of this simple and obvious precaution.

A careful and humane foreman can do much to mitigate the annoyance and suffering to which the workmen are subjected during the labor of tearing down a heap, by moving the point of attack from one to the other side of the pile, according to the direction of the wind, as well as by keeping the fresh surface on which the men are engaged well sprinkled with water to settle the fine ore-dust. At best, this labor is the most disagreeable and wearing connected with ordinary smelting, and, if possible, laborers should be changed periodically to some other employment.

Aside from the common tools already enumerated, long, stout steel gads and a few heavy sledges are needed to break up the central portion of the structure, which, although not fairly fused, is often so stuck together as to require considerable labor for its removal. At no other work are shovels so rapidly destroyed, and it is to this place that all partially worn, though still serviceable, tools are sent to terminate their existence.

The tearing down of the heap, and breaking-up of the matte that may be formed in it, are greatly facilitated by the use of a small quantity of dynamite, or other high explosives, selecting a powder of rather low force; containing not over 30 per cent. of nitro-glycerine.

When this is properly used and in not too large quantities, it saves infinite labor with bar and pick, a single shot, placed in a hole made in half a moment's time with a bar, often accomplishing more than hours of hard labor. The shot should simply shake up and loosen the mass, leaving the large lumps to be broken up by sledge and pick, as usual. If enough powder is used to break the whole mass up into small fragments, a great portion of the ore will soar into the air and go toward top-dressing the surrounding country.

I have never been able to get my men to be economical enough with their powder, except by forcing them to pay for it themselves. When they realize that every penny that is saved on powder goes into their own pockets, it is astonishing how little it takes to do the same work that required several times the quantity when it cost them nothing.

After the complete removal of the old heap, and any slight repairs that may be required to restore the ground to its former level, a thin layer of raw fines is again spread on the old spot, and the fuel arranged for a fresh pile. The estimate of costs for this process, as given below, is based on many different ores, varying greatly in composition, and under very various circumstances, and is purposely made somewhat liberal to allow for the occurrence of bad work and various other mishaps that are certain to occur in a greater or less degree. It is based upon a plant of 200 tons daily capacity, and on the assumption of only a short distance for transportation of the roasted ore to the smelting-furnace.*

* This is a considerable reduction on the original estimates for this process, as published in the previous edition of this work.

ESTIMATE FOR ROASTING 200 TONS ORE PER 24 HOURS.

Transportation by gravity-road at 4½ cents per ton	$9.00
Labor in building and burning heaps:	
6 men at $1.50 = $9.00	
2 men at $2.00 = $4.00	
	13.00
Five cords (640 cubic feet) wood at $5.00	25.00
Removing and loading roasted ore by contract at 12 cents per ton	24.00
One foreman	2.50
Screening, patching yard, etc., 2 men at $1.50	3.00
Oil, lights, repairs to cars, track, and tools, and new tools	11.50
Transportation to furnace in dump cars	9.00
Total	$97.00

Or $0.48½ per ton raw ore.

The various operations of heap-roasting may often be performed by contract to great advantage, especially if one has a good foreman to see that the quality of the roast is kept up to a satisfactory standard.

To give an idea of the prices that are fair for this operation, I will mention what I paid for roasting a heavy, pyrrhotite ore in large quantities, say 150 to 200 tons per day, the climate being excessively cold and stormy, and laborers' wages about $1.40 per 10 hours. The company furnished the wood for the roast-beds, and delivered the cars at the yard; but the cars had to be unloaded by hand and the raw ore wheeled to the heaps, the arrangements for dumping the ore direct not having been then completed.

For unloading the raw ore on the heaps, laying the wood, completing heaps, and covering and watching them throughout the entire operation, $0.22 per ton of ore.

For stripping, tearing down, and loading the roasted ore on cars, and unloading the cars by hand into the smelter-bins, $0.16 per ton of ore.

The contractors furnished their own powder, but the company provided tools, barrows, etc., though the contractors paid for the sharpening of their bars, picks, etc.

On the above basis, the contractors made a fair profit when they attended strictly to their business, and when there were no interruptions or shut-down. The degree of desulphurization arrived at by this process is seldom accurately determined, owing to the difficulty and expense of obtaining an accurate sample, and to the fact that the experienced eye can very correctly judge of the success of the roast, while any defect in the process will become immediately apparent in the lower tenor of the product of the succeeding fusion. Owing to the scarcity of accurate investigations on the subject, the following determinations were made:

No. 1. A heavy pyritous ore, from the Ely mine, Vermont, consisting principally of magnetic pyrites and chalcopyrite, burned in a heap of about 300 tons for eleven weeks. After stripping off the surface, a sample of the roasted ore, as delivered at the smelting-furnace, was taken. The following was the assay of the ore before and after calcination:

	Before Roasting.	After Roasting.
Sulphur	32.6 per cent.	7.4 per cent.
Copper	8.2 "	9.1 "
Insoluble	27.0 "	31.1 "

The condition of the copper in the roasted sample was also determined in this case, as follows:

Sulphate of copper	1.3 per cent.
Oxide of copper	2.1 "
Sulphide of copper	5.7 "
Total	9.1 "

No. 2. A heavy pyritous ore, being almost pure iron pyrites containing minute quantities of copper, silver, and gold, from the Phillips mine, Buckskin, Colorado, was roasted for 6 weeks in piles of 60 tons, and was used as a flux for siliceous silver ores. A careful sample of the roast yielded sulphur, before roasting, 46½ per cent.; after roasting, 11 per cent.

A considerable number of similar tests give corresponding results, showing that a very fair degree of desulphurization can be attained by this crude and ancient method, but still better results will be reached in ores containing less pyrites, and making the fact evident that, in heap-roasting as well as in the calcination of pulverized sulphides, the copper is the last metal present to part with its sulphur, and that a large proportion of this still remains in the condition of a sulphide after nearly the entire iron contents have become thoroughly oxidized. This agrees perfectly with all investigations relative to the comparative affinity of sulphur for the various metals, and is in no other metallurgical process more strikingly exemplified than in the so-called "kernel-roasting," as practised at Agordo, in Italy. There, the mechanical separation of the copper from its accompanying pyritous gangue is effected by stopping the process of calcination at the exact point where the entire iron contents have been oxidized into a soft earthy material, while the copper remains in combination with sulphur in a hard, metallic condition, and, most singularly, retreats into the center of each lump of ore, forming a heavy and solid kernel, which can easily be separated from its earthy envelope by inexpensive mechanical means. As this interesting process is not practised in this country, and in all probability is not suited to our domestic conditions, the student desirous of pursuing the subject will find further information in Plattner's *Röstprocesse*, as well as in a paper by the author in the *Mineral Resources of the United States* (A. Williams, Jr., 1883).

During the past few years, very much better results have been obtained in heap-roasting than would have formerly been considered possible. Ores containing over 40 per cent. sulphur are now often roasted down to 7 or 8 per cent., with regularity and certainty. This comes partly from longer experience of workmen, partly from premiums paid the men, based on the grade of matte produced in the subsequent smelting operation, and partly from using a much less quantity of wood to kindle the heap, and conducting the entire operation of roasting in a much more repressed and gradual manner.

The appearance of a freshly-opened heap of well-roasted ore is characteristic, athough difficult of description. It should present a strictly earthy, irregular surface of a blackish-brown hue, the scarcity of air preventing the oxidation of the iron to the red sesquioxide. This is a decided advantage in a reverberatory smelting-furnace, where the powerful carbonic oxide atmosphere of the blast-furnace is wanting to reduce it to the protoxide and thus fit it for entering the slag, the higher oxide being infusible at ordinary smelting temperatures. It is, in fact, principally a magnetic oxide, and, while the greater part of the contents should adhere closely together, and, when disturbed, should come out in the shape of large lumps, no sign of actual fusion should be visible, and the largest mass should fall into fragments at a few blows of the hammer. The more siliceous pieces of ore will have taken on a somewhat milky and opaque look in place of the ordinary vitreous appearance of quartzose minerals, and the veinlets of sulphides traversing the same will be found oxidized throughout. The solid lumps of pyrites, if carefully broken, will usually display a series of concentric layers, completely oxidized and earthy on the outside, and gradually acquiring greater firmness and a slight submetallic luster, which culminates in a rich kernel near the center of the fragment. This resembles strongly one or other of the grades of matte as produced from the smelting-furnace, and usually contains the greater part of the entire copper contents of the lump. The silver—if any be present—is also concentrated in a marked degree, though, so far as the author's own investigations extend, not with the same remarkable perfection as the less precious metal. The examination of a characteristic lump, such as just described, which contained before roasting about 4 per cent. of copper, yielded the following interesting results:

The outer earthly envelope contained....	Traces	of copper.
The medium concentric layers..........	1.2 per cent.	"
The central sub-metallic kernel..........	69.6 "	"

An imperfect roasting is quickly detected by the presence of more or less fused material at certain portions of the heap, while elsewhere there exists no cohesion between the lumps of ore, which fall apart like so many paving-stones. A certain metallic appearance will also be noticed, very different from the dull, earthy character of the properly burned pile. Although a large proportion of the contents may exhibit quite a brilliant red color, as though an unusually perfect oxidation of the iron had taken place, a mere weighing of one of the lumps in the hand will quickly undeceive the least experienced observer, and its fracture will show that the effect of the fire was only surface deep, while the entire interior remains unaltered. A careful study of different roast-heaps, wherever opportunity offers, will soon render the student skillful in judging by eye of the degree of success attained by this process, and in after-life frequently furnish him the key to the cause of the unsatisfactory tenor of the matte produced from his furnaces. No metallurgical process is more dependent upon an efficient and conscientious foreman, and the best results are usually obtained by selecting some intelligent and ambitious man from the roast-yard laborers, and holding him strictly responsible for results.

A decided improvement in heap-roasting of ores was introduced at the works of The Canadian Copper Company of Sudbury, Ontario, under the management of the author, in 1888–89. It was first tried by his assistant, Mr. James McArthur, and proved so valuable that it became a regular practice under ordinary circumstances.

We have called it the "V-Method" of roasting, and the accompanying sketch will make it clear. It consists in introducing a supplementary roast-heap between each two regular heaps, so that, if left untouched, there would be a continuous and unbroken roast-heap the entire length of the roast-yard.

The supplementary heap should not be built until its two neighbors, which are to form its lateral walls, are well under way, and have been lighted from 10 to 14 days. By this time, if properly managed, they will be cool enough on the outside to run no risk of setting afire the bed of wood which is laid down for the supplementary heap. The fresh bed of wood is laid down much thinner than for independent heaps, and a single layer is extended well up the slope of the two neighboring heaps. The ore is dumped on as rapidly as possible, and the heap finished off with ragging and fines in the usual manner, and fired from the ends.

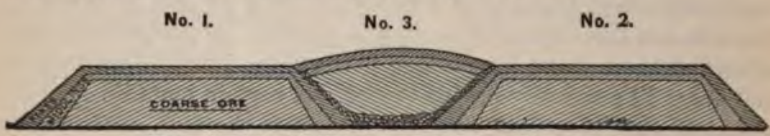


FIG. 19.—THE V-METHOD OF HEAP-ROASTING.

The result is excellent, for the new heap, having its sides protected, burns clear through its entire extent, and then sets on fire the still unroasted ore on the outside of the two neighboring heaps.

Thus the proportion of unroasted ore is reduced to a minimum, and indeed is seldom worth keeping separate.

Another great advantage is the economizing of space, for by this arrangement some 60 per cent. is added to the capacity of the roast-ground.

It may require some little patience and experimentation at first to adapt this practice to a new ore, but it is well worth the trouble, and has been pronounced by various members of our profession a decided and important improvement in this ancient and useful process.

In the case referred to, the ore that was roasted was a nickeliferous pyrrhotite mixed with chalcopyrite; but I have tried it sufficiently on both heavy and lean ores of the ordinary yellow iron pyrites to know that it is equally well adapted to all ores that are any way suited to heap-roasting.

HEAP-ROASTING OF MATTE.

There remains only, in connection with this portion of the subject, to notice the slight deviations that it is found necessary to introduce in adapting this method to the treatment of mattes.

These artificially formed sulphides, containing variable percentages of sulphur, may be sufficiently desulphurized in heaps, and their chemical composition has no marked effect upon the result, provided lead is not present to such an extent—15 per cent. or more—as to increase the fusibility of the material.

The most marked distinction between the behavior of ore and matte, when submitted to this process, is the fact that, while the former substance may be satisfactorily oxidized by a single treatment, the latter invariably demands two, and oftener three or more separate burnings, before it is properly prepared for the succeeding fusion. There is no exception to this rule, which, if properly understood, would prevent the disappointment frequently experienced by those unaccustomed to this method of desulphurizing matte and who are led to condemn the practice on finding, at the conclusion of the first carefully conducted burning, that the only visible results are a slight scorching of the surface of each fragment, a change in color from the original brownish-black to a brassy yellow, and a more or less extended fusion of such portions of the heap as have sustained the greatest heat. In reality, the influence of the process has been much more profound than can be realized from external appearances, and although neither the removal of the sulphur nor the oxidation of the iron and copper has progressed to any great extent, a certain change in the physical condition of every fragment of matte has been effected that prepares it perfectly for a second burning, and which seems a necessary preliminary to the actual desulphurization.

Each succeeding operation requires a slightly increased proportion of fuel, as the volatilization of the sulphur and the oxidation of the metallic constituents deprive the matte of its internal sources of heat, and at the same time greatly lessen its fusibility.

For the first roasting, a bed of wood should be prepared similar to that for a heap of ore, although smaller in area; for it is difficult to regulate the temperature and prevent matting in a heap much larger than 12 feet square, and this will be found a convenient size to hold from 60 to 70 tons of matte when raised to a height of about 6 feet. A single chimney in the center is sufficient, and about this structure the broken matte should be heaped just as it comes from the crusher or spalling-floor, and regardless of the fines that it contains. The presence of these has been found necessary to check the rapidity of the operation, and prevent the fire from suddenly spreading through the entire pile in a few hours without accomplishing any useful result, though generating for a short time a temperature high enough to fuse a large proportion of the contents into a single lump.

Less care need be taken in shaping a matte-heap than in the case of ore, and it is merely necessary to build it up in the form of a rude mound, which may best be covered with thoroughly burned ore from the roast-heaps, most of which on handling will crumble to a sufficient fineness for the purpose, while any hard lumps may be removed with the dung-fork. This obviates any screening or classifying of the matte in the open air, which always entails a heavy loss, owing to the great value and excessive friability and lightness of the material after calcination. If, as is usually the case, the proportion of fines after the first burning is found so great as to endanger the proper combustion of the heap for the second operation, the mechanical loss may be reduced to a minimum by separating the excess of pulverized matte by the use of a dung-fork, with tines closely set, during the turning of the ore from the heap just finished on to the fresh bed of wood, and at the conclusion of the process removing the fines that are thus isolated, either directly to the smelting-house, or, if they still contain too much sulphur, to the calcining-furnaces. The covering of the original heap, consisting solely of roasted ore, should be stripped off, and either sent to the smelting-furnace or again used for a similar purpose. It need hardly be mentioned that the presence of arsenic or similar impurities in the ore, in greater quantities than in the matte, should prevent any such practice as that just recommended, and it may be accepted as a universal rule in copper smelting, that no impure ores or products should ever be mixed with those freer from deleterious substances.

Under no circumstances need a matte-pile be covered as thoroughly as a roast-heap consisting of ore, nor can the formation of a considerable amount of matte, which in ore-roasting would be evidence of a great want of skill or care, be considered as a reproach, experience having so conclusively shown the impossibility of preventing its occurrence that, unless about one-eighth of the lower portion of a matte-heap is thus fused, no thorough oxidation of the remainder will be effected. The time necessary for the operations just discussed varies according to the quality of the matte, the condition of the weather, and certain other factors, but will in general be, for the first burning, eight days, while on the tenth day the heap will be sufficiently cool to permit its turning on to a fresh layer of fuel. The second operation requires a day longer, and the third a day less than the first burning.

To those familiar with the practice of heap-roasting as applied to ores, no particular directions are necessary except that care should be taken that the large blocks of matte that are formed during each burning be well broken up and placed near the center of the heap next constructed, that they may have every opportunity for a thorough desulphurization.

Whatever raw matte still remains from the last burning is best reserved until the construction of a fresh heap furnishes the proper means for its treatment. At the last two burnings, it is well to introduce two or more layers of chips, bark, or other refuse fuel into the matte-heap; for it will act powerfully in decomposing the sulphates that at this stage are formed in considerable amount, and also exercise a similar and most marked effect on whatever compounds of arsenic and antimony may be present. This simple measure had a sufficient effect in a certain instance in the experience of the author to be plainly noticeable in the quality of the ingot copper produced.

No attempt to select such portions of thoroughly calcined material as will be found after the second burning has ever proved remunerative. The heap of matte must be treated as a whole, and the roastings continued until the desired grade of desulphurization is reached.

The process just described is seldom an advantageous one, as, aside from the production of the vilest fumes known to metallurgy, the value of the material operated on is too great to admit of being locked up for 30 days or more, or to warrant the loss that necessarily results from such frequent handling in the open air. The last difficulty may be partially obviated by erecting a light structure to protect the heaps from the rain and wind; but, at best, the practice is an imperfect and objectionable one, and only to be recommended in new, outlying districts, where an expensive calcining plant cannot at once be erected, and where the climate is favorable for out-of-door operations. The expense of crushing and calcining in furnaces is decidedly less than the three or four burnings necessary to produce the same result; but the condition of the roasted material is so much more favorable for the succeeding smelting process, in the case of heap-roasting, that this reason alone is often sufficient to outweigh all objections that can be offered.

The practice of spalling the large pieces of matte upon the heap itself must be deprecated, as it has a strong tendency to solidify the structure and render the draught weak and irregular.

The cost of this process, based upon the roasting of many thousand tons of matte, and divested of those details that too closely resemble the heap-roasting of ore to warrant repetition, is as follows, assuming the daily amount of fresh matte subjected to this treatment to average 30 tons:

COST PER TON OF MATTE.

First Fire.	
Breaking	$0.19
Transportation to heap	0.06
Fuel—allowing 3 cords of wood to 60 tons of matte	0.25
Constructing heap and burning	0.21
Total	$0.71
Second Fire.	
Fuel—same as before with addition of chips	$0.30
Turning heap and burning	0.26
Total	$0.56
Third Fire.	
Fuel—same as second fire	$0.30
Removing finished heap	0.22
Transportation to furnace and expense of preparing the raw matte still remaining, which results from the fused matte	0.26
Total	$0.78
Total cost of three burnings	$2.05

STALL-ROASTING.

At just what period in the history of the art it became customary to inclose the roast-heap with a little wall of earth or mason-work, in order to protect it against the elements, to concentrate the heat, and to render unnecessary the tedious labor of covering the sides with fine ore, is unknown, though Agricola's work on metallurgy shows that it was no novelty in the sixteenth century. These simple walls have since been heightened and sometimes connected with an arched roof; the area that they inclose has been paved and occasionally furnished with a permanent grate; and, more important than all, the interior of the stall has been connected by a flue with a tall chimney, by which the draught has been improved, thus shortening the process of oxidation, while the noxious fumes are discharged into the atmosphere at such a height as to render them unobjectionable in most cases.

A very great variation exists in the size, shape and general arrangement of stalls, hardly two metallurgical establishments building them after the same pattern, though all essential differences may be properly considered by dividing them into two classes:

1. Open stalls, suitable only for ore.
2. Covered stalls, suitable for both ore and matte.

1. *Open Stalls.*—Any attempt at an exhaustive description of the different patterns of ore-stalls that human ignorance, as well as ingenuity, has invented, would be a waste of space. They all consist of a comparatively small paved area, surrounded by at least three permanent walls, and usually having an open front, which is loosely built up at each operation, to confine the contents. The back or sides, or both, are pierced with small openings communicating with a flue common to a large number of stalls that enters a high stack. The draught is confined to these passages by covering the surface of the ore with a layer of fines. From the great variety of existing patterns, one built at the works of the Parrot Copper and Silver Company, of Butte City, Montana, is selected for description as possessing exceptional advantages as regards cheapness of construction, convenience of filling and emptying, economy of fuel, and general adaptability.

The stalls may be built either of common red brick, of stone, or, far better, of slag molded into large blocks, which, from their size and weight, require little or no extraneous support; while brick demands thorough and extensive tying together with ironwork, and stone of proper size and shape is expensive and is apt to crack when exposed to great fluctuations of temperature.

As these so-called "slag-bricks" are invaluable for walls and foundations, and, in fact, for every purpose for which the most expensive cut granite would prove available, and as they can be produced from almost any copper slag that is not too basic, a brief description of the cheapest and best method of manufacturing them is appended.

MANUFACTURE OF SLAG-BRICK.

These are generally made from the slag of reverberatory smelting-furnaces, both because this material is usually more siliceous than any other, and also because, during the process of skimming, it can be obtained in large quantities in a very brief space of time. There should be no difficulty, however, in making the brick from the slag of a blast-furnace, provided the smelting is sufficiently rapid to fill the molds properly, and that it is not so basic as to yield too fragile a material on cooling. Even with exceedingly brittle blocks, produced from a highly ferruginous ore, excellent and durable walls can be constructed, provided the blocks are placed in position uninjured; for they will bear an immense crushing weight with impunity, and seem to defy the action of the elements.

Assuming the slag to be obtained from a reverberatory furnace, the process of preparing the molds should be begun as soon as possible after the slabs from the previous skimming have been removed and all chips and fragments cleared from the sand bed by the aid of a close-toothed iron garden-rake. Ordinary loam—or a natural mixture of fine sand and clay of such consistence that, when slightly moistened, it will ball firmly in the hand—is the proper material for the molds, which should be formed by means of a number of wooden blocks, of the required size, carefully smoothed and slightly tapered to facilitate their removal from the sand, and furnished with a 30-inch handle, inserted in their upper surface. These slag blocks are molded on the flat, in the same manner as ordinary red brick; and after leveling off the pile of dampened sand to form a smooth and horizontal bed, the wooden blocks—some twelve in number on each side of the skimming door—are arranged in a double row, four inches apart between blocks, and the same distance between the two parallel rows.

Besides the ordinary deep excavation for the plate slag, a second bed should be left on each side, between the former and the first brick mold right and left, both for the purpose of settling any grains of metal that may be accidentally drawn over during the process of skimming, and to act as a regulating reservoir to lessen the sudden impulse of the waves of slag that follow each motion of the rabble, and thus to prevent the destruction of the very fragile sand molds. The entire bed is constructed on an inclination of about one-half inch to the foot; the plate slag forming the summit, while the double row of molds slopes away from it in each direction laterally. After the wooden blocks have been placed on this sloping bed in a proper horizontal position, and exactly equidistant from each other, as determined by a wooden gauge, the remaining sand, very slightly but equably dampened, is shoveled back again, and carefully trodden and tamped evenly into all the interspaces and around the outside edges of the blocks, until it reaches the level of their upper surface. This is a very brief operation; for it is not essential to tamp the sand very firmly so long as about an equal degree of solidity is imparted to all portions of it. A cylinder of hard wood—3 inches in diameter and 4 inches long—which, when placed lengthwise, fits exactly between each two molds, is laid upon its side, and, by a few blows of the mallet, driven into the sand, thus when removed forming a little gutter through the middle of the partition wall, and connecting each pair of adjacent cavities in such a manner that the flow of slag through either entire lateral system meets with no impediment. The wooden blocks are then removed from their sand bed with the greatest care, it often being necessary to loosen them by gentle tapping and other means familiar to the experienced molder. The bed requires only a few hours' drying to fit it for the slag.

By the time the charge is ready for skimming, say in three hours or less after the completion of the bed just described, it should be in proper condition, and the furnace helper, armed with a small rabble-shaped hoe, stands beside the skimmer ready to turn the stream of slag into the proper molds, remove obstructions from the gutters, br ak through the rapidly forming crust if indications of chilling appear on the surface of the molten bath, and see in general that the process of filling the molds proceeds in a proper manner. As soon as this operation is concluded, a few shovelfuls of sand should be thrown over the surface of the slabs to prevent sudden and unequal chilling. By the time the new charge is in the furnace and the assistant is at liberty to attend to his bricks, they will usually be found ready for removal, though still at a red heat on the surface and in most cases quite liquid in the interior. It is essential that they be removed, and the slight roughnesses that arise from the broken ends corresponding to the gutters through which they were filled be trimmed off with a small cutting hammer while they are still quite hot, as it is just at this stage that they possess the highest degree of toughness, and permit of manipulations that, if they were cool, would inevitably break them into fragments. These slabs are best removed from the furnace by being loaded upon the low iron barrow commonly used for the transportation of pigs of slag and matte. The loading is effected by means of a long five-eighths inch iron rod, bent into a hook at one end, and the blocks are then wheeled out upon the dump, where a special workman trims them properly, rejecting all that are imperfect or already cracked, and, when cool, piles them into rows, to remain until needed. The most useful size for general purposes has been found to be about 8 by 10 by 20 inches, and weighing about 85 pounds; but by simply changing the form of the pattern, they may be produced of any desired shape or size, although experience has shown that it is not economy to attempt the manufacture of very thin slabs, or of any weight below 45 pounds. The immense value of this building material, produced from an otherwise worthless substance and obtainable in rectangular shape for plain walls and foundations, in wedge shape for arches and for forming a circle in walling wells and for many other daily needs, can be fully appreciated only by those who have had occasion to build in a country where rock was unobtainable and brick poor and expensive.

The slow cooling, or tempering, of slag will greatly increase its toughness and strength, but it is only in late years that this method has been applied to the manufacture of slag brick from the basic, ferruginous slags of ordinary blast-furnace work. The following description, with illustrations, is taken from Dr. Egleston's paper on "The Manufacture of Slag Brick in Montana," *The School of Mines Quarterly*, Vol. XII.

The process which is used at the Parrot Works at Butte, Montana, was invented by Mr. J. E. Gaylord, of that company, and is interesting because it allows of quickly arriving at the result with ordinary labor, and is applicable anywhere and to almost any slag, provided it holds together on cooling, as almost all the slags in the West do. It consists simply in dumping the fluid slag from the inside of the ordinary conical iron pot into a cast-iron mold, instead of allowing it to get cool in the pot and then throwing it on to the dump heap. This requires that the casting yard shall be near the furnaces, so that the slag-pots shall not have to be wheeled too far, and that the space shall be large enough for the men to work conveniently, and also space for the storage of the hot molded slag while cooling. The plant required for this manufacture is of the simplest description, and the product available

SLAG BRICK MOULD

Parrot Works

BUTTE, MONTANA.

PLAN

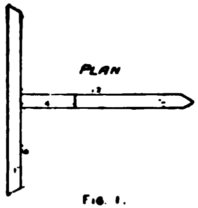
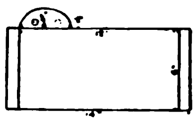


FIG. 1.

CAST IRON USED FOR MAKING MOULD

ELEVATION



SCALE

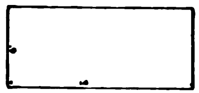


FIG. 2

CAST IRON BED PLATE

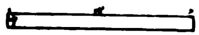
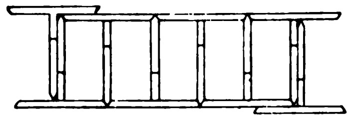


FIG. 3.

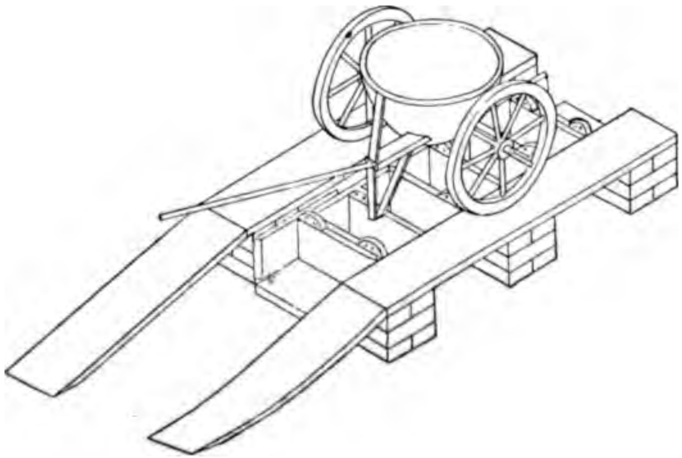
MODE OF FORMING MOULD



SCALE

FIG. 4.

SKETCH SHOWING MOULDS IN USE



for almost any building required about the works, or, indeed, for any ordinary construction, especially for underground work.

The slag-bricks at these works are made by contract, and are paid for at the rate of 85 cents to $1 per hundred. The bricks are 12 inches long and 6 inches wide and high. This has been found by experience to be the most convenient size, but they might be made of any other size when it was desirable to do so. The Parrot brick weighs about 55 pounds; one man can make 350 in a day. They are made on an area near the blast-furnaces. At these works there are two plants for making them, each plant having three sets of apparatus at a distance of about 30 feet apart. These three sets are worked by one man in the day and one in the night shift; or four men in 24 hours. When there is a greater demand, extra sets can be easily set up or shifts of eight hours can be made. The apparatus consists of a set of cast-iron plates, shown in the accompanying plan and elevation. These plates are cast in the shape of a T and have beveled ends. They are one inch thick, 14 inches long, 12 inches wide on the inside and 14 on the outside. The bevel occupies one inch, so that the available inside space is 12 inches long and 6 inches wide and high. This piece is set upon a series of bed-plates, which are 14 by 6 and 1½ inches thick. These are leveled up and form a floor, and are juxtaposed so as to leave their joints under the mold frames. The frames are placed together, so as to form five molds, so that the pointed, beveled ends of the long end of the T fit into the V-openings made by placing the beveled ends of the short ones together. The method of placing them is shown herewith. No special end pieces are made for the purpose of resisting the pressure, but two of the castings are placed at the end for that purpose. On the outside of these and resting upon supports, 9 inches high, plates of cast-iron 6 inches wide and of the same thickness are set, a little longer than the length of the five molds. They are reached by an incline three feet long, placed as shown in the cut, so that the wheels of the slag-pot will run on them and be just over the molds below. The incline is so gentle that there is no difficulty in pushing the slag-pot up it. The pot full of slag from the furnaces is run up this incline. The man shoving it makes two holes in the crust, which has cooled on the top while coming from the furnace, the front one to pour the slag out and the one on the other side behind it to allow of the flow. He then tips the pot over by raising the handle of the slag-wagon, and the melted slag on the inside falls into the molds below until they are full. There will then be a shell of slag on the inside of the pot. This is carried to the dump-heaps and tipped there, and the pot is taken back to the furnace to be again filled. By the time the molds in plant No. 3 are full, the brick man, who has just prepared these molds and has watched the operation of casting, is ready to take to pieces the bed No. 2 previously cast. He goes there, and with a hook which fits into the holes on the top of the castings, shown in the elevation, pulls out the irons and puts them to one side, leaving the hot bricks on the iron pavement to cool sufficiently to be handled. When this is done he goes to No. 1, the bricks of which have been cooling and are ready to be piled, but are still hot. He takes them up on a shovel and piles them close together, making headers every other row. He then reconstructs the molds in No. 1, putting the irons, which are still hot, in place by means of the hook, washes them with clay water, and by this time the bricks of No. 2 are ready to be piled. He first goes to No. 3, pulls out the irons and then piles the bricks of No. 2, and by this time fresh slag comes to No. 1, and so on. It does not take much more than ten minutes between the casting of one set and the making of the piles of the other. The bricks are left in the pile until they are quite cool, by which time they are sufficiently annealed to be used.

There is always a considerable quantity of small stuff, made by the slopping of the slag. This is taken away by one man with a horse and cart, and is used for making the roads about the works and for filling either between masonry or in the ground. These bricks are constantly used about the works, and considerable quantities of them are sold to be used in the town. They are very advantageous for construction, as they require less mortar than ordinary bricks, and are quite as strong as stone, when they are not liable to shock. They are used exclusively in the construction of the kilns at that works, where they would last a very long time, but for the habit of cooling down the hot ore with water, which makes it necessary to reconstruct them every four or five years. The bricks of the size made here are the most convenient. If made smaller they would cost too much, since the labor would be about the same whatever the size If made larger they would be too heavy for the men to handle conveniently. They can be transported short distances and are cheaper and more easily laid than stone.

The skill of the manufacture is entirely in keeping the irons above ground, moving them frequently and keeping them coated with clay water. When, as in some cases, the molds have been permanently fixed in place and the slag allowed to cool in them, the cast-iron pieces have become useless in a short time. At the Parrot Works, where the work is done carefully, they last indefinitely, and where the molds are taken to pieces as soon as the bricks are strong enough to hold themselves up, the wear is inappreciable. The process is a very ingenious and simple one and applicable at any works producing slag. The cost of the plant is very small, the labor required is not high-priced, and over two-thirds of the slag is a source of a small profit to the works, instead of being an incumbrance and a source of expense.

ROAST STALLS FOR ORE.

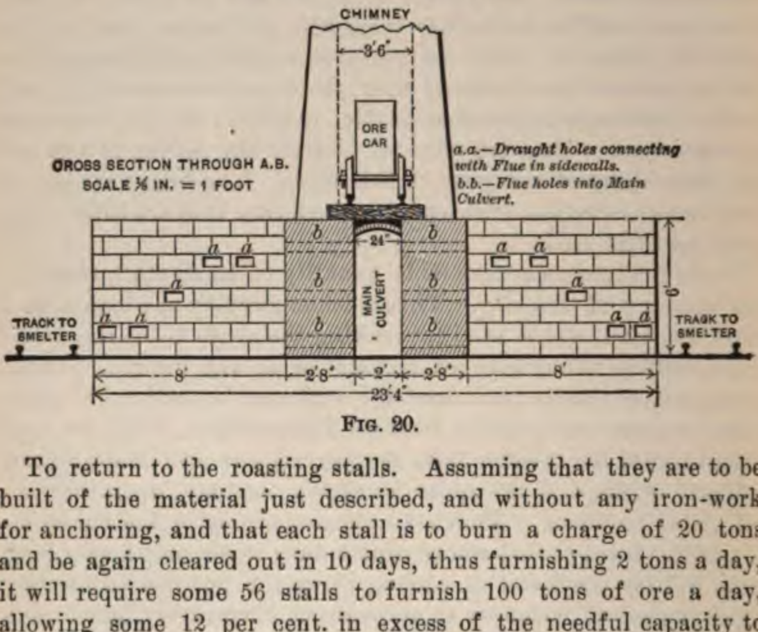


FIG. 20.

To return to the roasting stalls. Assuming that they are to be built of the material just described, and without any iron-work for anchoring, and that each stall is to burn a charge of 20 tons and be again cleared out in 10 days, thus furnishing 2 tons a day, it will require some 56 stalls to furnish 100 tons of ore a day, allowing some 12 per cent. in excess of the needful capacity to permit of repairs. The weight of ore as brought to the stalls, and not as *taken from* them, is counted: its loss during the process of calcination depends upon the quality and amount of sulphides present, and frequently reaches 15 per cent., though a considerable portion of the loss in weight due to the elimination of the sulphur is offset by the gain in oxygen.

Such a battery of stalls should always be built in a double row, back to back, each lateral wall serving as the division between the two adjacent partitions, while the unbroken rear walls form the sides of the main flue, a space of at least two feet being left between them, which simply requires a 4-inch brick arch to form the main flue for the entire system. This also constitutes a foundation on which, after a little leveling up with earth, to prevent the sleepers from being affected by the heated masonry below, the narrow railroad is laid on which the ore for roasting is brought to any part of a given stall by means of the turn-plate and movable rails, explained in the chapter on "Heap-Roasting." A double row of 28 stalls (56 in all) should have a flue at least 2 by 4 feet for the third of the number nearest the chimney, which may be reduced to 2 by 3 feet for the middle, and 2 by 2½ feet for the end third, if any saving can be effected thereby. The two long rear walls, enclosing the main flue, should be 32 inches thick—once and a half the length of a slag-brick—with proper allowance for mortar and irregularities, and should be laid solely in clay mortar, which designation throughout this entire work may be understood to mean merely common sticky mud, such as is employed for making a poor quality of red brick. If ordinary clay be accessible, it may be mixed with sand in such proportions as to slip easily from the trowel; otherwise, any ordinary sticky mud may be used, and will be found to form perfectly satisfactory material for laying all mason-work that is to be exposed to sulphur fumes and a heat not exceeding a dull red.

The fact that lime mortar is totally unadapted to ordinary metallurgical uses, although doubtless universally known, is for some unaccountable reason frequently not acted upon, and the result in most cases is the rapid and total destruction of the furnace-arch, chimney, flue, or whatever structure may happen to have been put together with such unfit material. The acid vapors immediately form a sulphate with the lime present in the mortar, and this, absorbing water, becomes gypsum and crystallizes, expanding with great force, breaking the joints, and soon crumbles and washes away. It is quite proper to use lime mortar in such portions of the structure as are free from contact with heat and sulphurous gases, and yet require unusual strength, which cannot be obtained with the clay substitute. Such, for instance, as in the construction of chimneys for metallurgical purposes, where the best results can only be obtained by the employment of both of

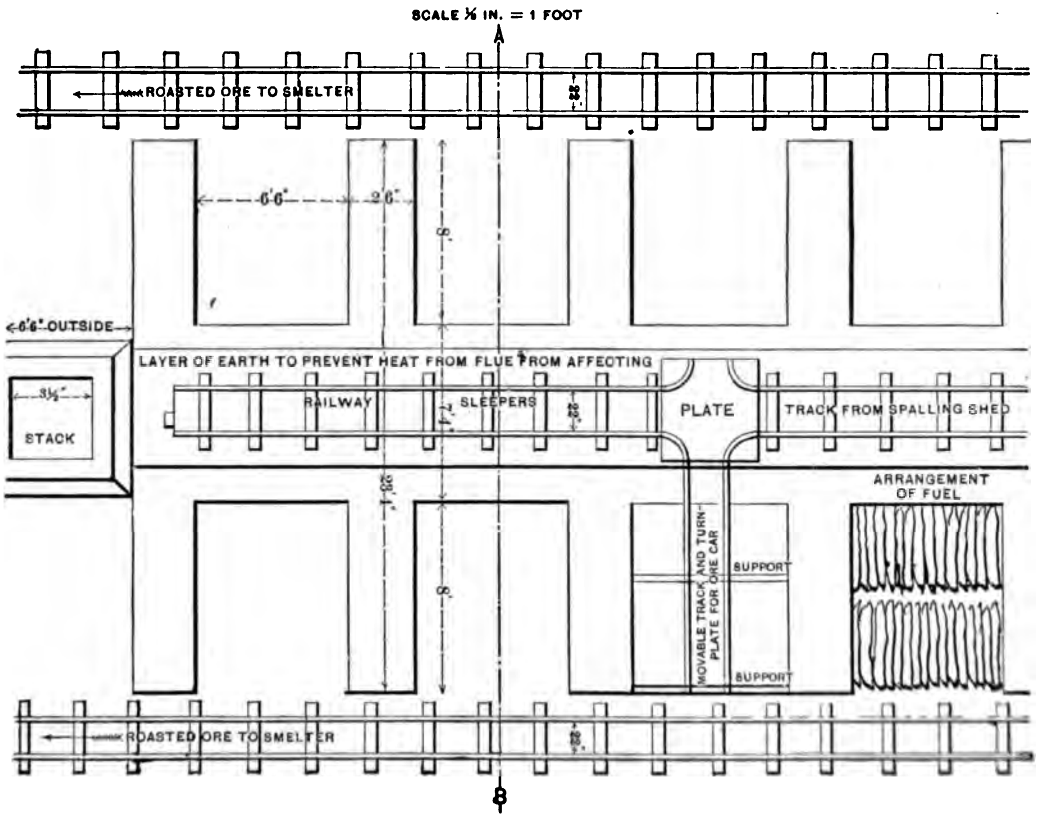

SCALE ⅛ IN. = 1 FOOT
ROASTED ORE TO SMELTER
6'6"
2'6"
8'
6'6" OUTSIDE
LAYER OF EARTH TO PREVENT HEAT FROM FLUE FROM AFFECTING
3½"
STACK
RAILWAY
SLEEPERS
7'4"
PLATE
TRACK FROM SPALLING SHED
ARRANGEMENT OF FUEL
23'4"
MOVABLE TRACK AND TURN-PLATE FOR ORE CAR
SUPPORT
SUPPORT
8'
ROASTED ORE TO SMELTER

FIG. 21.

these substances: lime mortar for the outside work, while the common clay mud is merely used for the inside layer, and the joints thoroughly protected against any invasion of the sulphur gases by plastering the whole interior with a thin coating of clay mortar, tempered with sand to such an extent that it will not crack and fall off in sheets. Further reference will be made to this point in speaking of "Furnace Building." The constant and flagrant violation of this law is a sufficient reason for its frequent reiteration. A recent example suggests itself, where the arches of a number of very expensive and nearly new calcining-furnaces had fallen in, causing a very heavy loss. A conversation with the mason who built them brought out the fact that they were constructed with lime mortar, he having had no orders to the contrary.

The size of the stall is somewhat dependent upon the quality of the ore to be roasted, a highly siliceous ore with a comparatively low percentage of sulphur permitting a much wider and higher stall than an ore with little gangue, and especially than one containing a considerable portion of iron pyrites, in which case extensive and unavoidable sintering will follow any attempt at increasing the size of the stall. A safe size for an average ore, containing a moderate amount of pyrite and demanding careful handling, is 8 feet in length by 6 feet in height by 8 feet in width. It is best to build the lateral walls of the same thickness as the rear division, the increased stability and durability of the entire structure well repaying the slight additional expense in labor and material. The bottom should be paved with the same slabs placed flatwise and exactly reversed from the position in which they lay when formed; their upper surface now going downward, while their original lower surface, which should be perfectly smooth and level, now comes upward. The connection with the main flue is effected by means of 8 or 10 small rectangular openings—2 by 6 inches—in the rear wall, in two or more rows, and at a considerable distance from the ground. These are kept tightly closed by means of a bunch of old rags or a ball of clay, when there is no occasion for their remaining open; otherwise, the draught of the entire system might suffer.

The only air admitted to these stalls originally, at the Parrot works, came through such interstices as were left in roughly building up the temporary front wall; but experiments led to the addition of some 4 or 6 similar openings in each lateral wall, which did not communicate with the main culvert, but connected with the outside air by means of a small flue running longitudinally through each division wall, though not extending so far as the central passage. This innovation has been followed by a decided improvement in the oxidation of the ore and a great diminution in the amount of matte produced. An essential precaution in the management of these stalls is to maintain a thick coat of clay plastering over their entire interior surface, by which the heated ore is kept from sticking to the walls and causing the rapid destruction of the mason-work. A few moments' attention to the empty structure after each operation will keep the plastering intact and greatly lessen the cost of repairs. As the entire success of this process depends upon the strength and regularity of the draught, a stack of considerable size and height is essential.

A battery of 56 stalls, as described, requires at sea-level a chimney 75 feet high, and with an internal area of at least 12 square feet, as will be further explained in the chapter on the construction of calcining-furnaces. Any economy in the direction of diminishing the size of this important adjunct will be immediately noticed in the lengthening of the roasting process, and may reduce the capacity of the stalls to an incredible degree. The draught is regulated by means of a sheet-iron damper hung in the main flue, close to its junction with the chimney, while the same office is accomplished for individual stalls by partially filling the draught-holes in the rear wall with bits of bricks or balls of clay. In no portion of the process are the skill and care of the roasting foreman better displayed than in his management of the draught, which must be varied according to the season and temperature of the air, as well as with the changing character of the ore.

As already intimated, a stall of the size and pattern just described will hold 20 to 30 tons of pyritous ore, which should be kindled with the very smallest possible quantity of wood that will set it thoroughly on fire. This is essential for a far more important reason than the mere saving in fuel; for the slightest increase in the contents of the bed of wood on which the rock is heaped will, with pyritous or otherwise easily fusible ores, cause an amount of sintering and a formation of matte entirely disproportionate to the cause. Repeated trials can alone determine the various minutiæ of this description essential to the best possible results with the material under treatment; but, in most cases, where the ore is at all pyritous, good sound wood will be found to produce too fierce a heat for the purpose, and recourse must be had to decayed wood, which can usually be obtained at from one-half to two-thirds of the price of the sound fuel. For an ore containing 30 per cent. sulphur and, say 25 per cent. silica, 25 cubic feet of rotten wood, or about one-fifth of a cord, will be found ample; but this small proportion of fuel—only one-hundredth of a cord to the ton—must be utilized in a proper manner, and with the most rigid economy and exactitude, or the heap will miss fire completely, doubling the cost of the operation, as well as interfering with the estimated production of the plant. A quarter of an hour spent in watching the manipulations of an experienced roaster is better than pages of description, though the operation of preparing a stall for its ore charge is far from complicated.

After seeing that the layer of clay on the enclosing walls is renewed with the plastering-trowel where necessary, and that the draught-holes are open to the extent dictated by former experience, a central longitudinal, and two lateral flues are constructed on the floor of the stall out of large, irregular fragments of ore. These are merely to introduce air to the interior and to insure the rapid and thorough kindling of the entire structure. They are filled and surrounded with dry kindling-wood, and the greater part of the fuel, split into long, thin sticks from the large rotten logs and poles that are usually provided, is disposed in a thin layer over the bottom of the stall, the amount slightly increasing toward each side. The structure is now filled with coarse ore, and the ragging distributed throughout the entire contents rather than concentrated in a considerable layer merely upon the surface. As the stall becomes gradually filled, single small sticks of wood are placed between the ore and the lateral and back walls; while between the contents of the stall and the front wall, which is built up with large lumps of ore or stall matte, a considerable quantity of light wood is introduced to insure the thorough desulphurization of the anterior surface. A single carload of ragging is spread on top of the coarse ore, and upon this a three-inch layer of shavings, bark, and chips is placed as a bed for about one and a half tons of raw fines, which, if disposed in the exact manner indicated, and covered closely with a well-roasted ore from a contiguous stall, will be thoroughly desulphurized, and the covering layer itself being in a well calcined condition, the entire contents, after burning, may be passed on to the next operation. Mr. R. Pearce, of Argo, uses with great advantage a sheet-iron cover over the top of his stalls, luted tightly with clay to the walls on which it rests.

It is only by employing great care, and after repeated trials, that the requisite skill will be attained to thoroughly calcine the large proportion of fines just indicated; but when one reflects that it amounts to some 7 per cent. of the entire ore, and perhaps one-half of the total fines produced, it will be seen that the result is worthy of any pains that can be expended on it. The large pieces of raw ore that are employed in building the flues and front wall become gradually oxidized upon the surface, and slowly crumble away and mix with the finished product until they totally disappear and are replaced by fresh pieces. When the ore is to be removed, the front wall is taken down, and the lumps of ore from it are piled out of the way, to be again used for the same purpose.

The stall should be fired at night, as the smoke is so dense during the first few hours, and the draught so sluggish, that only a small part of the fumes find their way into the proper channel; but by the time the wood is consumed, the entire structure has become so much warmer as greatly to improve the draught. The sulphur and other products of volatilization and "sweating"—alluded to in describing the management of roast-heaps—form a sort of crust upon the surface, and seal all interstices connecting with the atmosphere, and force nearly all fumes to pass into the flue, thus greatly abating a nuisance. For the first twenty-four hours, the fire is confined to those portions of the ore that were in immediate contact with the fuel. The process of oxidation advances very rapidly, and by the close of the second day it is hardly possible to bear the hand upon the middle of the upper surface of the stall, showing that at least one-half the contents is already in combustion. By the end of the fourth day a similar degree of temperature may be felt upon the upper surface, at the very back of the stall, proving that the process has by that time invaded the entire length and breadth of the stall, though considerable time is still necessary for its thorough completion.

The successful progress of the process is clearly marked by the great rise in height of the entire contents, gaining some three inches in a single day, and frequently ascending some 12 inches above the level of the walls at which it stood at the beginning of the operation, aside from the free space left to be filled out with ore from the disappearance of the fuel, amounting to some 25 cubic feet. This striking phenomenon, unfamiliar to those accustomed only to heap-roasting, where a settling rather than a rising of the entire mass occurs, is simply due to the fact that, in all cases of oxidizing roasting, a greater or less, though always very marked, increase in bulk occurs from the swelling and fissuring of the oxidized ore. The contents of the roast-heap, being perfectly free and unconfined, simply spread out laterally, while the consumption of the thick bed of fuel on which it rests detracts considerably from its height. The walls of the stall, however, enclose the ore in a rigid grasp, making it absolutely necessary that any increase in bulk, beyond that very slight amount necessary to replace the space occupied by the fuel, should take place vertically. In a badly burned stall, where extensive sintering has taken place, and a sufficient amount of the sulphides has melted into a solid mass to cause a decided diminution in bulk instead of an increase, the occurrence of crater-like depressions in the surface of the ore is positive evidence of such local fusions. That the pressure resulting from the increase in bulk is something quite tangible, may be inferred from the frequent pushing outward, or even overturning of the heavy lateral walls of a stall, provided one or the other of its contiguous compartments is either empty or unbraced, while the temporary front wall would inevitably be thrown down within the first day after kindling if not strongly supported by timbers.

The length of time necessary for the process under consideration is another uncertain factor. If the stall be left undisturbed, it will usually burn quietly for a period of twelve days, demanding little or no attention beyond an occasional shovelful of covering if heating too fiercely at any one point, and requiring about three days additional to cool sufficiently to remove with comfort; but, under ordinary everyday circumstances, no such moderation can be practised, and the period of each operation can be curtailed, without any especial damage, to one-half this time. To accomplish this without detriment to the process of desulphurization, the following precautions must be adopted: As soon as the anterior surface of the ore is so cool as to impart no disagreeable sensation to the hand, the temporary front wall should be removed, the natural adhesion common to all sulphureted ores when roasted in lumps preventing the caving of the vertical ore face, which should be most carefully attacked with pick and shovel, every precaution being taken not to penetrate beyond the line of comparative cooling, and only so much ore being removed at any one operation as is consistent with the uninterrupted progress of the roasting in the mass behind. At least six or eight inches of ore should be left between the outer air and the line of fire, and any sudden elevation of the surface temperature, as well as increased difficulty in detaching the ore from the face on which work is prosecuted, is a sign to stop. To illustrate the ease with which the contents of a well-burned stall can be handled, the entire charge of ore from such a stall can be removed with nothing stronger than a shingle.

The first carload is usually taken from the stall at the close of the fourth day, and the amount capable of removal may be rapidly increased, until in seven days more the compartment is again empty.

By this careful method of constant and systematic slicing, some two or three tons of well-burned ore may be taken daily from each of 40 or 50 stalls, and the capacity of the roasting plant rendered more than double what it would be if they were left untouched for the time necessary for their complete desulphurization and cooling; while the process of oxidation does not suffer in the slightest degree if the precautions just enumerated are adhered to.

In the case of ores containing arsenical pyrites, or, indeed, in the presence of any form of arsenical or antimonial combinations, a considerable proportion of the same that would otherwise go into the next operation in the shape of antimonates and arsenates may be volatilized and completely dispersed by the admixture of chips, small coal, brushwood, or other carbonaceous materials, which, as in heap-roasting, exercise a powerful reducing influence upon the products of oxidation just mentioned, and volatilize them in a metallic form. This simple precaution is of much greater value in the calcination of similar compounds in a pulverized condition in furnaces, where the different periods of oxidation and reduction are under the control of the operator, and can be made to follow each other in the manner most conducive to the object in view; but even in the rude process under consideration, experience has shown, in many cases, that a decided improvement in the grade of copper has resulted from this device, the simplicity and economy of which are among its strongest recommendations.

The results obtained in stall-roasting vary little as compared with those from burning in heaps. On the whole, it is not quite so easy to prevent the formation of matte in the former practice, nor do average and oft-repeated examinations show quite as good results in the elimination of the sulphur.

As circumstances may arise where it becomes the duty of the constructing metallurgist to decide between these two systems, to the positive exclusion of all methods involving the pulverization of the ore, and to give his reasons for and against each method, that his employers may also have some idea of the matter on which to base their advice or to rest the confirmation of his decision, it will be well to concisely review the comparative advantages and drawbacks of heap and stall-roasting.*

The first and most obvious advantage of the system of heap-roasting is the apparent cheapness and simplicity of the plant, only a level area being required, without furnaces, flues, stacks, or other expensive appurtenances.

The extreme simplicity of the method and the very satisfactory results obtained under proper management also speak in its favor; but further than this no arguments can be advanced in support of the process.

Even the economy in first cost of plant will be found more apparent than real, when the expense of the trestle-work and track, as well as the establishment of the different grades between spalling-shed, roast-yard, and smelting-house levels are considered, and no one will deny the absolute necessity for such an arrangement if work on a large scale is to be prosecuted with any degree of economy.

A careful comparative calculation of costs, corrected by the results of actual work, shows that, under ordinary circumstances, the difference in cost between the two plans under consideration is too trifling to have much weight in the choice of methods, and may even be on the side of the stalls in cases where the natural conformation of the land is unfavorable for the establishment of the terraces necessary for cheap heap-roasting.

A far more important reason for the adoption of the stall system is the great saving in time, by which the delay incidental to the cruder process of calcination is diminished by at least 80 per cent.

In works of large capacity, this becomes a question of vital importance; for few smelting companies are so amply provided with capital as to carry a constant stock of some 10,000 to 50,000 tons of ore, representing a money value of several hundred thousand dollars, which is not at all an extravagant estimate for works of the capacity under consideration. The circumstance that this amount

* See article on "The Mines and Smelting-Works of Butte City," by the author, in the United States publication on *Mineral Resources* (by A. Williams, Jr., 1885). The third method of roasting lump ore—that is, in continuous kilns—is only suited to certain peculiar conditions, and need not be considered when comparing the other two systems.

may be reduced to a sum not exceeding one-fifth of the above by the substitution of the quicker method of calcination is a weighty argument for its adoption.

A still further advantage may be claimed for them in the concentration of all noxious fumes into a single flue, and their discharge into the atmosphere at such an elevation as to insure their gradual diffusion and dispersion without annoyance or damage. This is a great boon to the surrounding country, and more especially to the workmen employed in the process of roasting, as any one familiar with the atmosphere of an establishment where heap-roasting is practised can testify.

Still further may be mentioned the considerable saving effected by the thorough roasting of the entire contents of the stall, including even the fine covering material, all of which is in condition for the succeeding operation; whereas, in the case of heap-roasting, at least 10 per cent. of the entire stock requires a second handling. Here may also be considered the serious losses of metal from wind, rain, and other atmospheric causes, which, although not entirely obviated by the employment of stalls, are at least greatly lessened; the saving in a certain plant of moderate capacity amounting in a single year, according to the author's calculations, to more than sufficient to cover the entire cost of erecting the stalls.

But the most important advantage possessed by stall-roasting over heap-roasting in an ordinarily moist climate—if the process be carried on in the open air—is the prevention of loss by leaching.

The cost of stall roasting will not vary far from 50 cents per ton of ore, with the same prices as are assumed in the estimate for heap-roasting.

The cost of a battery of 56 stalls, built in the manner recommended and reduced to the standard table of price adhered to throughout this work, is appended. Their life, under ordinary treatment, will not exceed six years, at the expiration of which time they will be found in such a condition as to demand complete rebuilding, although, of course, the stack will outlast several generations of stalls.

ESTIMATED COST OF 56 ROASTING-STALLS, EXCLUSIVE OF STACK.

This being the first estimate yet given pertaining to the construction of any considerable portion of a smelting plant, the quickest and most convenient method of arriving at the desired result will be presented a little more in detail than may be consid ered necessary in subsequent calculations.

The total expense of the finished stalls may be conveniently divided into the following heads:

1. Excavation for foundations.
2. Cost of slag-brick, clay, and other building materials, delivered on the ground.
3. Labor in building the stalls.
4. Total expense of the railroads belonging to this part of the plant.
5. Miscellaneous expenses and superintendence.

The actual expense of building a plant of this description will almost invariably be found much greater than the most carefully prepared estimates would indicate, unless the figures were made by a man of long experience in these matters. The value of the numerous estimates of cost and expense contained in these pages is principally due to the fact that they are, almost without exception, taken from the results of actual work, executed under the superintendence of the author. They may, consequently, lay claim to a usefulness and reliability that the most carefully prepared estimates of cost would not possess unless derived from, or at least corrected by, a long and thorough personal experience in such matters.

To prepare the foundations for the required number of stalls, assuming the ground to be comparatively level, will require about 60 days' labor, aside from the removal of the earth. This allows for an 8-inch pavement, and for an extension of the foundation walls about two feet under ground.

1. *Excavation for foundations:*

Labor, 60 days at $1.50	$90.00
Removing the excavated material	35.00
Superintendence and miscellaneous extras	32.00
Total	$157.00

In order to estimate the amount of building material required, it is essential to determine the cubic contents of all the walls inclosing the 56 stalls, 28 in each row. The stalls being 6½ feet wide, and all walls being 32 inches thick, it will be seen that the entire length of the two main rear walls is 520 feet, to which must be added the aggregated length of the 58 partition walls, each 8 feet long, or 464 feet, making a grand total length of 984 feet.

This wall being 6 feet high and 32 inches thick contains in round numbers 15,700 cubic feet. To this must still be added about one-third, to allow for the foundation walls, and also the necessary amount of slabs for paving the stalls. The details are as follows:

Main walls	15,700	slag-brick.
Foundation walls	5,250	"
Paving	2,080	"
Total	23,030	"

As these slabs are 8 by 10 by 20 inches, they contain very nearly a cubic foot each, and when the very coarse joints that they form are also considered, it will be found that their customary rating of a cubic foot each will be perfectly safe. They are laid entirely in ordinary clayey loam, which may be found almost everywhere, and which, if too sticky to leave the trowel, will be greatly improved by the addition of one-fourth or more of sand, or even sandy loam. At our standard of prices, $1 per ton will be ample for such material, and will lay one hundred brick. The cost of the slag-brick has been placed at two cents on the ground, as their delivery is at least as expensive as their manufacture. The sum mentioned, that is, two cents apiece delivered, or one cent at the furnace, will cover the cost of making and trimming, and leave enough margin to occasionally replace the pattern blocks and other material necessary for their production.

2. *Cost of materials for mason-work.*

23,000 slag-brick, at 2 cents	$460.00
235 tons clay, at $1	235.00
Mortar-boxes, hods, screens, etc.	45.00
Total	$740.00

The persons employed for this work should on no account be the regular, high-priced brickmasons, as these fare but badly in handling the heavy, brittle slabs, and neither like the work nor are able to earn the large wages that they invariably demand and receive. The proper mechanics for this work are what are popularly known as "country stonemasons," whose apprenticeship at building stone walls, underpinning barns and houses, etc., has exactly prepared them for handling such rough and heavy material as that under discussion.

Experience in this particular kind of construction has shown that the most advantageous distribution of the force is to provide each stonemason with two immediate helpers, who assist him constantly, bringing the slab, placing it in position, and, in fact, doing everything excepting the spreading of the mortar and that last wedging and chinking that are of such vital importance in the proper execution of work of this description.

There are no hodcarriers, as the slabs are delivered by wagons at the point most convenient to the workmen, and the mortar, easily and rapidly manufactured from the materials already mentioned, is brought in large pails, being used in immense quantities in work of this description, although every crevice should be well filled with small fragments of rock or slag, called "spalls."

It has been found that each group of three men, as described above, will lay on an average 100 slag-brick daily, and not more.

3. *Labor in building stalls.*

Estimate for laying 100 *brick:*

One stonemason	$3.00
Two laborers at $1.50	3.00
Mixing mortar for same	.50
Carrying mortar and other miscellaneous labor	.15
Superintendence	.35
Total for 100	$7.00
Total for 23,000 brick	$1,610.00

4. *Cost of Railroad Tracks.*—As all railroads about the works should be of the same gauge and pattern, a single detailed estimate will determine the cost per foot once for all. For tracks of the required description, having a 22-inch gauge, and calculated to carry a net load of 1,800 pounds, the car weighing an additional 800 pounds, a good quality T-rail of not less than 12 pounds to the yard should be selected and well fastened in place by a spike in every sleeper, while the abutting ends of the rails should be firmly secured by fish-plates, tapped for four ⅝-inch bolts, two to each rail. Unless the bolt-holes in both fish-plates and rails can be bored where ordered in such a manner that there shall be no doubt of their perfect correspondence, it is better to leave the plates blank, and bore them on the spot. This may seem a slight matter, but its neglect sometimes causes serious annoyance and delay in outlying districts, and the boring of the thin fish-plates is a slight task, as every smelter should be provided with a boring-machine run by power, which is indispensable for sampling pig-copper, and will be found generally useful.

The sleepers are sawed from the ordinary timber of the country, and may be conveniently ordered of the following dimensions: 36 inches long, 6 inches wide, and 4 inches thick—containing each 6 feet, board measure. They should be placed 39 inches apart from center to center, and last almost indefinitely, as the sulphate salts with which they become impregnated prevent their decay.

For convenience of calculation, the estimate will be based on a length of 100 yards of track:

Weight of iron:

200 yards rails at 12 pounds = 2,400 pounds.		
Spikes, bolts, and fish-plates = 115 "		
Total........ .... 2,515 pounds × 3½ cents =		$88.02

Sleepers:

125 containing 6 feet each = 750 feet at $25 per M =		18.75

Labor:

Grading, laying track, ballasting, etc.	$13.50	
Superintendence	6.00	
Total		19.50
Average allowance for curves and switches		13.63
10 per cent. for incidentals		14.00
		$153.90

We may therefore accept the figure of $1.53 per yard, or 51 cents per foot, as the cost of a tram-road of this description, and there being three lines of track required for the stalls, aggregating a length of 780 feet, to which must be added 100 feet for connections, switches, and single main line to smelter, we have a total of 880 feet at 51 cents, or $448.80.

Rails for long curves may be bent cold; for short curves, they must be slightly heated; while frogs, points, etc., require welding, and can be readily constructed in any ordinary blacksmith's forge.

Great care should be taken in laying the track, nor should the foreman rest satisfied until every point, frog, and guard rail is in proper position and has the precise curve necessary for easy passage of the car without undue friction or danger of derailment. It is scarcely necessary to say that this work can only be properly and economically executed under the direction of an experienced railroad constructor.

5. *Miscellaneous Expenses and Superintendence.*—Aside from the allowance made in each department of the work for the above

purposes, it will be found in practice that a considerable additional sum is required to cover errors in construction, blacksmith work, and various incidentals, as well as general superintendence, amounting in a case similar to the above to

	$211.00
Cost of 4-inch brick arch to cover main flue	137.00
	$348.00

Summary.

Excavation for foundations	$157.00
Materials for mason-work	740.00
Labor in building stalls	1,610.00
Railroads	448.80
Miscellaneous and superintendence	348.00
Grand total	$3,303.80

Uneven ground, bad weather, and other unfavorable causes may increase this sum to a considerable extent, but the figures given will be found safe under ordinary circumstances and with strictly judicious and economical management.

The calcination of matte in ore stalls of the pattern just described is by no means impossible, the principal difference between its treatment and that of ore being the increased quantity of fuel required—about three times as much. A considerable proportion of the matte will be fused during the operation, and another large fraction scarcely affected by the process; so that from three to four burnings are required to effect any reasonably perfect desulphurization.

This practice cannot be recommended, as much better results are obtained by providing the stalls with grate-bars, and preventing the radiation of heat from the surface by means of an arched brick roof.

THE STALL-ROASTING OF MATTE.

This is a method well known in the Eastern States, and practised first in this country, so far as any record can be found, at the old Revere Copper Works in Boston, and in more modern times at Copperas Hill in Vermont, and at the noted Vershire mine in the same State, where some sixty or seventy stalls are still in use. The partial suppression of the excessively disagreeable fumes generated in the heap-roasting of this substance; a gain of at least one-third in the time of treatment—no unimportant item in the handling of such valuable material; and a very great diminution in the losses caused by the elements are the principal reasons for the selection of stalls in preference to heaps. On the other hand must be placed a heavy investment in buildings and in the stalls themselves, with their flues, stacks, etc. The mere grate-bars for a single matte stall cost in the neighborhood of $75, and the constant repairs that are peculiarly necessary in the case of mason-work saturated with the products of volatilization, and racked by the frequent and extensive fluctuations in temperature, due to the ever-recurring heating and cooling of the interior, render them a somewhat expensive portion of the plant, as will be seen in detail in its proper place.

MANAGEMENT OF MATTE STALLS.

The grate-bars being thoroughly cleansed and freed from all clinkers and *débris* of the preceding operation, and replaced in position, and the brick walls forming the sides and back of the stall receiving a fresh coat of plaster (clay) where necessary, a layer of fuel is placed upon the grate-bars, and the broken matte thrown upon this by means of a closely-tined fork, to separate the fine stuff, which is scattered over the top after the stall is filled with an average charge of from five to six tons.

The fuel employed is wood in 4 or 6-foot lengths, and split to a comparatively uniform size. From 10 to 20 cubic feet are used for each charge, metal of low grade, rich in sulphur, requiring less fuel than the higher varieties of matte. Experience has taught the great advantage obtained by the use of hard wood, and too much care cannot be bestowed upon the selection of the fuel, which should be of the best quality and thoroughly seasoned, as the result of the operation depends to a remarkable extent upon the quality of the fuel used.

Matte of any grade, from the lowest coarse metal to the highest quality of regule, may be treated in these stalls with almost equal results as regards desulphurization.

The stalls are always covered by rude sheds, to protect the brick-work from the weather, and should be paved with slag blocks, flat stone, or, much better, heavy iron plates, as the constant hammering that it must undergo during the spalling of the matte and the breaking of the huge clinkers that form an almost necessary accompaniment of this process, quickly destroys any other description of pavement. The results of desulphurization by this method being no more thorough than by heap-roasting, the same number of burnings is necessary as in the latter case, and, owing to the difficulty of removing the heavy clinkers from the walls and grate-bars of these little furnaces, as well as the constant bill of expense for repairs, the cost of the process is about the same as in heap-roasting. The almost complete identity of the two methods in this respect renders any further details of expense unnecessary. The imperfections of all the methods of roasting matte in lump form, as well as the great waste of time and metal, and the annoyance caused by the fumes, are serious objections, and it is only under exceptional circumstances that these crude and dilatory methods can be recommended. In nearly all advanced works, they have given place to the much more rapid and perfect method of calcination in reverberatory furnaces.

The ordinary dimensions of the stalls in use, now or formerly, at some of the principal works in this country are as follows:

Width	5 feet.
Depth (front to back)	6 feet.
Depth of ash-pit	1 foot 6 inches.
Height from grate to spring of arch	4 feet 8 inches.
Thickness of division walls	1 foot 4 inches.
Thickness of rear walls	1 foot 8 inches.
Area of flue opening in rear wall	160 square inches.

A stall of this size will contain from five to six tons of matte, and will burn for four days at the first firing, and for about three days at each subsequent operation.

Where three burnings take place, the capacity of each matte stall may be placed at one-half ton daily, and the amount of wood required for the three burnings will be one-twelfth of a cord per ton of ore.

From the measurements already given, aided by the estimates for brick-work found in a succeeding chapter, the cost of a block of such covered stalls may be easily arrived at; the covering arch consisting of a 9-inch semicircle of red bricks, and the main flue section being at least equal to the combined area of the flues that enter it.

The anchoring of a block of such stalls is very simple, consisting of longitudinal ¾-inch rods, while the uprights may be iron rails or stout wooden timbers. Each side wall should also be braced from front to back in the usual manner, while the front wall of the stall is a temporary structure of brick laid loosely upon the grate-bars and braced with a few lengths of flat iron. Fire-brick are ordinarily used for this purpose, the common red brick of which the entire permanent portion of the structure is built being too light and fragile to stand the repeated handlings and the fluctuations of temperature.

Since the ordinary charge only fills the stall about two-thirds full at the front, and slopes up against the rear wall to nearly the height of the flue opening near the top of the walls, or even in the arched roof, a large space exists between the upper edge of the temporary front retaining wall and the high semicircular brick roof. Through this, the sulphurous fumes and the products of the combustion of the fuel during an early stage of the process escape in such clouds as to render the atmosphere of the shed unfit for respiration. To partially obviate this difficulty, a sheet-iron curtain, suspended by wires running over a pulley in the roof, and furnished with a counter-weight, is used, and if properly fitted and luted to the side walls with a paste of stiff clay, is of great service.

It may be assumed with safety that, by the process of matte-roasting in lump form—whether executed in heaps or covered stalls—from two-thirds to three-fourths of its original sulphur contents is eliminated, by not less than three consecutive burnings.

THE ROASTING OF ORES IN LUMP FORM IN KILNS.

By the term kiln, as used here, we understand a comparatively small, shaft-like furnace, provided with a grate or opening for the admission of air from the bottom, and connected with a draught flue. The action is a continuous one, and the necessary heat is derived entirely from the oxidation of the sulphur and the other constituents of the ore.

No other class of furnaces has received greater attention or been brought to a greater state of perfection; but it is as an adjunct to the manufacture of sulphuric acid rather than to the calcination of ore that this apparatus must be esteemed, and consequently to the works treating on that subject that we must look for detailed descriptions and estimates of the same. The student is referred to Lunge's exhaustive work on "Sulphuric Acid" for full details of construction and management.

While the various processes of roasting hitherto described are suited to almost every variety of sulphureted copper ore, and yield equally good results whether the percentage of sulphur and copper is small or large, a much closer selection of material is indispensable for successful roasting in kilns, and their range of usefulness is restricted to comparatively narrow limits.

This very question of selection, however, varies greatly with the purpose in view, and depends upon whether it is desired merely to desulphurize a given ore without any attempt to utilize the volatile products of oxidation, or whether the manufacture of sulphuric acid is to be combined with the process of roasting.

The conditions necessarily present before any pyrites can be utilized for the manufacture of sulphuric acid are of two kinds, commercial and technical.

The commercial conditions are sufficiently obvious to any thoughtful mind, and are very plain, such as sufficient supply of ore at a fixed and low rate for a reasonable length of time, and contiguity to water, railroads, or some cheap means of transportation to the manufactory, which, owing to the nature of its product, must be situated in the immediate vicinity of its market.

The technical conditions, though more numerous, are almost equally easy of comprehension. An almost absolute freedom from gangue is essential, for the simple reason that the presence of foreign substances lowers the percentage of sulphur and necessitates the handling of worthless material, thus lessening the capacity of the works and producing other unfavorable results. For the same reason, though in a less degree, the presence of any other sulphides but the bisulphide of iron, which forms the ore proper, is disadvantageous; for no other compound of sulphur contains either so high a percentage of the same or parts with it so freely. Even the copper pyrites, which in many instances forms the principal value of the ore, is detrimental to the manufacture of sulphuric acid, both because it contains less sulphur and because it is too fusible to permit the proper regulation of the temperature. Beyond the limit of 8 per cent. of copper in the pyrites, it cannot be profitably employed in the manufacture of acid. The Spanish pyrites, from which so large a proportion of the acid produced in England is made, contains on an average about 3 per cent. of copper, and about 48 per cent. of sulphur, this remarkably high percentage of sulphur showing its freedom from gangue.

An analysis of the average ore from the celebrated Rio Tinto mine may be of interest, as a type of a very favorable cupriferous pyrite for acid making:

ANALYSIS OF RIO TINTO PYRITES BY PATTINSON.

Sulphur	48.00	Magnesia	0.08
Iron	40.74	Arsenic	0.21
Copper	3.42	Insoluble	5.67
Lead	0.82	Oxygen and moisture	1.00
Lime	0.21		
Total			100.15

The ore used by several large acid-works in Boston and New York is obtained principally from Canada, some thirty miles from the Vermont line, and although somewhat variable in purity, averages about 3.5 per cent. of copper and 45 per cent. of sulphur, the percentage of gangue being greater than in the Spanish ores.

An excellent quality of pyrites is mined from a large deposit in Western Massachusetts, and in both Virginia and Georgia are beds of pyrites now under process of development, which, on competent authority, are said to rival the Spanish mines in quality.

The presence of arsenic and antimony has a deleterious effect on the quality of the resulting acid, while lead heightens the fusibility of the charge, besides wasting sulphur by forming a stable lead sulphate, and any foreign substance, however harmless otherwise, lessens the percentage of sulphur.

An important point, sometimes overlooked by non-professionals in determining the value of a sample of pyrites, is its mechanical behavior during the process both of crushing and of roasting. A granular ore, soft or easily disintegrated, will increase the proportion of fines, which, although now utilized with great success in the manufacture of acid, are still undesirable as requiring a more expensive plant and entailing a greater cost in their treatment. A still more serious production of fines may take place in the kiln itself in the case of ores that decrepitate, sometimes occurring to such an extent as entirely to choke the draught and render their employment impossible.

One of the most serious errors ever perpetrated in the manufacture of acid from pyrites is the attempted employment of pyrrhotite, or monosulphide of iron, for pyrite—bisulphide of iron. Aside from the greatly lessened proportion of sulphur, 36 per cent. as against 53 per cent., the monosulphide will not even yield freely what sulphur it contains, but crusts with oxide of iron, turns black, and is soon extinguished when treated in an ordinary pyrites kiln. It seems scarcely possible that extensive works for the manufacture of sulphuric acid (and copper) should have been erected, their ore supply being entirely derived from a deposit of the valueless monosulphide; but such has been the case in more than one instance, and will continue to be so in enterprises conducted without the aid of skilled direction. One of the most striking instances of this kind is a now extinct Massachusetts company, which is said to have expended over $200,000 in this manner, all of which was a total loss, excepting the small amount realized from the sale of buildings and land.

Under certain conditions, the use of kilns for the calcination of cupriferous pyrites without the production of sulphuric acid may be found advantageous, as in the case of the former Orford Nickel and Copper Company, near Sherbrooke, Province of Quebec, which, after employing heap-roasting for some time, erected a large number of kilns solely for the purpose of calcining its ore previous to smelting; finding the saving in time and avoidance of waste, combined with the lessening of the annoyance formerly experienced from sulphur fumes, a sufficient advantage to repay the somewhat heavy cost of the burners.

The minimum percentage of sulphur sufficient to maintain combustion in kilns does not yet seem to have been positively determined; but with an ore otherwise favorable, it is probable that 25 per cent. is quite sufficient for the purpose.

For economy's sake, as well as for the purpose of retaining the heat, kilns are constructed in blocks of considerable length and of the depth of two burners, the front of each facing outward, while the flue in which the gas is conveyed to its destination is built on top of the longitudinal center wall. Fire-brick are used wherever the masonry is exposed to heat or wear, and the entire block of furnaces is surrounded by cast-iron plates, firmly bolted in position, and provided with the necessary openings for manipulation.

No fuel is required after the burners are once in operation; and when in normal condition, the attendance demanded, aside from the labor connected with the regular charge of from 500 to 2,000 pounds of ore once in twelve or twenty-four hours, is very slight.

Much skill and experience, however, are required to maintain the regular working of the kilns, especially with ores that are not exactly suited to the process.

From 5 to 10 per cent. of fines may also be desulphurized with the coarse ore without seriously interfering with the process. They are thrown toward the back and sides of the shaft, leaving the center uncovered; otherwise, the draught is affected and serious irregularities supervene.

In accordance with the policy adopted throughout this work, no detailed estimate of expense will be given in the few instances where the author is unable to base the same on personal experience. Such is the case in kiln roasting; but we are assured by the best authorities that the expense of calcination by this method does not exceed that of stall roasting, though the first cost of the plant is considerably greater.

The results obtained by this process are unexampled in the roasting of lump ores, although there is no doubt that a considerable share of the success is due to the fact that the sulphur is the object of interest, instead of merely being a waste product to be driven off as far as convenient.

If more than 4 per cent. of sulphur remains in the *cinders*, as the residue from this process is called, the result is not considered satisfactory. It is needless to say that such a perfect desulphurization cannot be obtained in either heap or stall-roasting, nor is it necessary or, in many cases, even beneficial for the subsequent process, although, of course, in most instances the lack of sulphur in the furnace charge forms a welcome outlet for the admixture of raw fines, which may thus escape the expense of calcination.

Within the past few years, the utilization of these fines has attracted much attention, and the efforts to calcine them in automatic furnaces for the production of sulphurous acid have been crowned with success, as will be again alluded to when treating of the "Roasting of Pulverized Materials."

The attempt to utilize kilns, with certain slight modifications, for the roasting of copper matte has, after many difficulties and much expense, attained a successful issue at certain European works, especially at the Mansfeld copper works in Germany, the object in view being rather the abolition of the nuisance arising from the escape of the sulphur fumes into the atmosphere than any expectation of financial advantage from the employment of a substance so poor in sulphur for the manufacture of acid.

CHAPTER VII.

THE ROASTING OF ORES IN PULVERIZED CONDITION.

AT the beginning of Chapter V. we classified the apparatus suitable for roasting ores in a finely divided form, as follows:

1. Shaft-furnaces.
2. Stalls.
3. Hand reverberatory calciners.
    (*a*) Open hearth.
    (*b*) Muffle.
4. Revolving cylinders.
    (*a*) Continuous discharge.
    (*b*) Intermittent discharge.
5. Automatic reverberatory calciners.
    (*a*) Stationary hearth.
    (*b*) Revolving hearth.

SHAFT-FURNACES.

This group includes some of the most important and useful appliances for the roasting of sulphureted substances, where the utilization of the fumes for the manufacture of sulphuric acid forms a part of the process of calcination.

If the question of acid manufacture be left entirely out of consideration, and the comparative economy of each method of calcination be judged solely upon its own merits, it is doubtful whether resort would be had to these furnaces, save under exceptional conditions, as their limited capacity, great cost of construction, and imperfect work, except under the most skillful management, would effectually bar their introduction. But under the stimulus arising from the enforced manufacture of acid from pulverized pyrites, and the consequent necessity of employing some form of automatic furnace in which the gases arising from the oxidation of the ore are kept separate from the products of combustion of the fuel, this type of calciner has received such attention and study that it promises fairly to rival the most economical form of roasting apparatus known to metallurgy. The student is referred to Lunge's work on the manufacture of sulphuric acid for full details regarding this and other forms of furnace suited to the calcination of ores in connection with acid-making.

The Gerstenhofer shelf-furnace was the first successful calciner of this type, and is still largely used, though becoming gradually supplanted by improved modifications. The few furnaces of this pattern that have been constructed in the United States have failed to answer the desired purpose, owing to imperfect construction, poor refractory materials, and want of skill in management. The Gerstenhofer furnace consists of a vertical shaft, surmounted by a mechanical device for feeding the pulverized sulphides in any desired quantity, and contains a great number of parallel clay ledges of a triangular form, one of the flat surfaces being placed uppermost. These are so arranged as to obstruct the ore in its passage, and delay it sufficiently to effect a certain degree of oxidation, which is seldom perfect enough to yield the desired result without a supplementary calcination in some other form of furnace. The front wall of the shaft is pierced by parallel rows of rectangular openings, for the purpose of changing the clay shelves or of cleansing the same.

The oxidation of the sulphides generates sufficient heat for the proper working of the process, so that the sulphurous gases may be obtained for the manufacture of acid free from any products of the combustion of fuel.

Though greater capacity has been reported, I have never been able to satisfy myself that a full-sized Gerstenhofer could handle more than 6,000 pounds per 24 hours of granular pyrites, with 46 per cent. sulphur, the residues averaging 6 per cent. sulphur.

Hasenclever's furnace consists of a vertical shaft containing six to eight inclined fire-clay shelves arranged in a zigzag fashion, as in Stetefeldt's dry kiln. The angle of inclination is about 40 degrees, so that a thin layer of ore covers each shelf, descending by its own gravity as rapidly as the finished product is carried out at the bottom by a fluted roller. The discharge is continuous, but the capacity very limited, seldom reaching one ton of fines per 24 hours. Nor can the heat be maintained without extraneous aid, which is usually supplied by connecting it with a kiln in which lump pyrites is burned.

The Malétra furnace properly belongs under "Hand Reverberatories," as it consists of a number of small hearths one above the other, over which the ore is moved by hand-rakes. This furnace is interesting as roasting fines entirely without the aid of extraneous heat. It will roast about one ton of heavy pyrites fines per 24 hours down to 3 per cent. sulphur.

The Stetefeldt furnace, so invaluable for the chloridizing-roasting of silver ores, is a shaft provided with a grate for the generation of such a degree of temperature as would be lacking in the roasting of ores so poor in sulphur as those usually exposed to this treatment, as well as an auxiliary fireplace for the more perfect chloridizing of the flue-dust, which, owing to the fine pulverization of the ore and the strong draught essential to the proper working of the apparatus, is formed in unprecedented amounts, and pretty thoroughly regained in ample dust-chambers.

The employment of an auxiliary fireplace, and the invention of a highly ingenious and perfect automatic ore-feeder, constitute important claims to originality that are frequently overlooked by writers in commenting on this furnace. Its capacity is very great, 60 tons in 24 hours being easily worked in one of the large-sized furnaces of this type; and were it possible to obtain equally good results by employing it for oxidizing-roasting, it would be the most valuable addition to the modern metallurgy of copper. But as it is at the present time, it cannot be enumerated among the resources of the copper smelter, although late experiments indicate the probability of its successful adaptation to this purpose.

STALLS.

Pelatan's roasting and agglomeration furnace presents some novel features, and is intended for the calcination of pyritic smalls or other sulphides. It consists of a long, narrow covered stall, provided with sheet-iron front, and cast-iron side-doors. It contains a close-meshed grate. The charge of fine or granular ore, after being placed in the grate, is ignited from below. A light blast is used under the grate, and it is claimed that while the plant is cheap, the capacity is large, and that a 10-ton charge of most ores can be well roasted and slightly agglomerated in 12 hours. If such results could be constantly obtained in practice, the apparatus would be of much value in many places. Good results are reported from the Laurium galena mines in Greece, and from pyrites mines in Chili, that are working up their old piles of fines in this furnace.

HAND REVERBERATORY CALCINERS.

(*a*) *With Open Hearth.*—The essential features of the ordinary reverberatory calciner are a hearth, heated by a fireplace, from which it is ordinarily separated by the bridge-wall, and accessible by certain openings through the side walls, the whole being covered by a flat arch against which the flame *reverberates* in its passage from the grate to the flue, thus being brought momentarily in contact with the ore spread upon the hearth, while the combined gases from fuel and charge pass into the open air through a chimney, in many cases first traversing a series of flues and chambers for the purpose of retaining such particles of metal as may have been either chemically or mechanically borne away by the rapid draught.

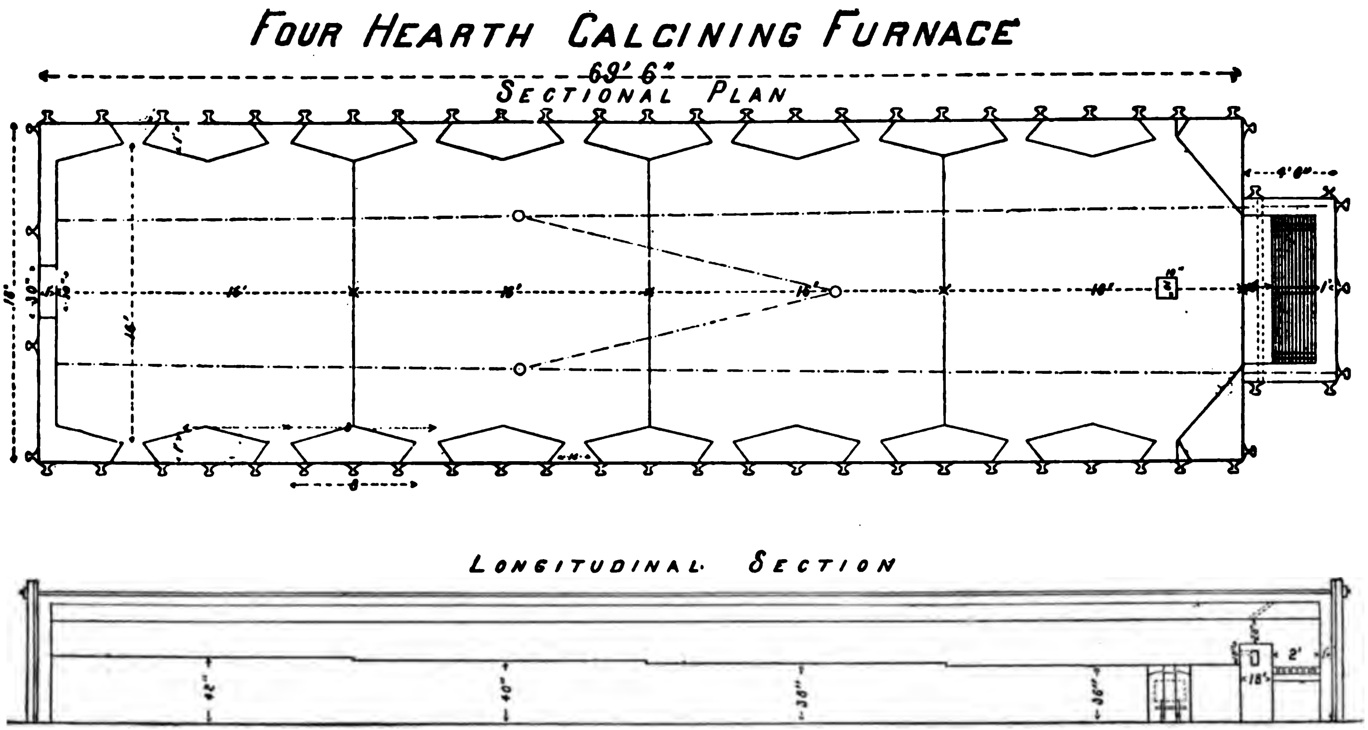
A very small grate surface, as compared with the hearth area, distinguishes this type from the reverberatory smelting-furnace, and corresponds to the very moderate temperature suited to the process of calcination, permitting its almost entire construction of common red brick.

A single detailed account of the longest and largest variety of calciner in common use will serve as a model for all smaller specimens of the same class.

The principal variable dimension of a copper-desulphurizing furnace is its length, as, for economical reasons, its width should always be as great as is compatible with convenient manipulation. Experience has placed this limit at 16 feet for the inside measurement of the hearth, nor should this dimension be lessened without good and sufficient reasons.

The length of the hearth is limited chiefly by the capacity of the ore to generate heat during its oxidation, the immediate influence of the fireplace being seldom capable of maintaining the requisite temperature upon a hearth over 35 feet in length, without resorting to the use of a forced blast, or of a draught so powerful as greatly to increase the loss in dust as well as the consumption of fuel.

The importance of the heat generated by the oxidation of sulphides in maintaining a proper temperature, and especially in conveying the heat to a great distance from the initial point, is not always appreciated. Its intensity and durability depend upon the percentage of sulphur in the ore, and also not a little upon the manner in which it is chemically combined, the bisulphides—such



FIGS. 22 AND 23.

as iron pyrites—furnishing a much greater amount of heat than monosulphides containing an equal gross amount of sulphur.

An ore carrying less than 10 per cent. of sulphur will not furnish sufficient heat to warrant the addition of a second hearth to the first 16 feet, which will be assumed as the normal length of a single hearth. (Such a condition would scarcely occur in practice, as, under ordinary circumstances, any copper ore containing such a low percentage of sulphur could be smelted raw.) An increase of sulphur to 15 per cent., however, will be sufficient to heat the second hearth, while a 20 per cent. sulphur ore should work rapidly in a three-hearth furnace. The addition of a fourth and final section is rendered justifiable by the increase of the average sulphur contents of the ore to 25 per cent., and even a 20 per cent. bisulphide charge may be worked to advantage in the same.

The adoption of this method of roasting, by which the ore is fed into one end of the furnace, and gradually moved to the other extremity before discharging, is attended with several obvious advantages, among which are: The gradual elevation of temperature from a point compatible with the easy fusibility of the unaltered suphides to that degree necessary for the complete decomposition of the pertinacious basic sulphates of copper and zinc; the great saving in fuel effected by thus obtaining the full benefit of the heat generated in the process of roasting itself; the certainty that the charge must undergo a certain number of thorough stirrings and turnings in its transportation over so extended a space; the establishment of a fixed duty, which must be performed by the workmen, whose labor can thus be much more easily controlled than with the single-hearthed type of calciner, where the attendants can easily substitute an idle scratching for the vigorous manipulation necessary to move the ore forward promptly; a great simplification in firing, it being only necessary in the long furnace to maintain an even, high temperature, while with the single hearth, much experience and judgment are required to adapt the heat to the ever-varing condition of the charge; lastly, a decided economy in construction, the ratio of fire-brick to common red brick for an equal capacity of plant being much less in the employment of long furnaces.

As there seems to be almost no limit to the extent of surface over which the requisite temperature may be obtained in the calcination of highly sulphureted ores, it is very natural that experiments should have been made with still longer furnaces than any yet mentioned, 120 feet being the extreme inside length yet attempted, so far as known to the writer; but careful and repeated trials have shown beyond a doubt that no sufficient advantage is reaped to pay the increased cost of the enclosing building and other expenses of plant. It is not possible for two attendants properly to manage a furnace having more than four full-sized hearths, if the latter is pushed to its full capacity, while the addition of a fifth hearth demands a third laborer, whose time, however, will not be fully occupied, while a sixth hearth will overtax the three workmen. In short, the testimony of many excellent metallurgists, to which the author can add his own experience, unequivocally condemns the lengthening of ordinary calcining-furnaces beyond the limits above indicated, excepting under special and peculiar conditions.

The number of working-doors to a long calcining-furnace, where the ore is moved from rear to front, should be as few as possible. The limit for comfortable work should not exceed 8 feet between centers of doors, and any distance less than 6 feet is a decided disadvantage.

The sides of the working-door frames should have short lugs, not exceeding 6 inches in length, cast on them, in order that they may be firmly held in position by the buckstaves, which are placed in pairs for this purpose, a single buckstaff being placed in the center of the space between each pair. The bottom of the door-frames should be on a level with the hearth surface, which should be 3 feet above the floor grade of the building, and should slope gradually upward toward the rear of the furnace, to correspond with the increased height of each succeeding hearth.

The common practice of filling up the portions of the hearth between the working-doors with projecting, triangular masses of brick-work cannot be recommended, as valuable space is often sacrificed in this manner. Slight projections, as shown in the cut, may be built to fill the absolutely inaccessible angles; but with properly constructed door-frames, and careful manipulation on the part of the roasting attendants, but little waste area should exist, and this will regulate itself by becoming filled with ore, which may remain there permanently. This refers, of course, to the treatment of large quantities of low-grade ores, where slight inaccuracies resulting from the trifling mixture of ores can do no harm.*

* These building directions are, in the main, equally applicable to any of the automatic calcining furnaces.

After raising the side walls to the height required by the iron door-frames, usually about ten inches above the hearth level, the skewback for the main arch should be laid. This applies to the entire furnace from the beginning of the fire-box to the extremity of the rear hearth, and is a very simple matter, especially if the arch is to be perfectly horizontal, as is to be recommended in most cases. A taut line should be stretched, to insure accurate work, and if red brick are used, they should be cut on one long edge, being laid, of course, longitudinally and on the flat. They should be cut at an angle slightly greater than required by the curve of the arch, which should rise about three-quarters of an inch to the foot, making a 16-foot arch 12 inches higher in the center than at the sides. This rise, though less than is often recommended, will be found ample to insure perfect safety and durability, and will tend to spread the flame and heat toward the sides of the hearth.

If so-called "side skewback" fire-brick are within reach, they should be used in place of the red brick, saving much cutting and insuring a better job. Three rows, in height, of red brick, or two of fire-brick, will give a solid bearing, the total number required for a furnace of the size under consideration being, respectively, 600 and 375.

It is of sufficient importance to bear repetition, that all portions of the mason work above the hearth line, or wherever exposed to heat, must be laid in clay—common brick clay, tempered with sand, being quite good enough for all portions of the furnace—as fire-clay is usually expensive in the localities where copper ores abound.

Lime mortar, much improved by the admixture of a little good cement—say 10 per cent.—may be advantageously employed for the outside work and wherever there is no danger of heat, as it makes handsomer and stronger work, and is greatly preferred by the masons, who require constant supervision to compel them to use clay mortar where it is necessary.

The heavy iron bridge-plate, so indispensable in reverberatory smelting-furnaces, may be entirely omitted, the bridge being built up solid and covered on the top and sides with fire-brick, with the exception of a longitudinal opening 3 by 8 inches, which should penetrate it from one end to the other, communicating with the outside air on each side of the furnace, and with the hearth by some dozen 2 by 4-inch openings.

By this means, heated air free from all reducing gases is admitted into the furnace below the sheet of flame that sweeps over the top of the bridge. The oxidizing effect of this current of air is very powerful, and, as frequently determined by experiment, hastens materially the calcining process.

If wood is used as a fuel, an additional row of similar openings should be constructed in the arch, immediately over the front line of the bridge-wall, by which a much more perfect combustion of the gases is effected. With coal as a fuel, the latter openings are less urgent, provided the firing is properly managed.

Aside from the 16 working-door frames, and the ordinary doors for fire-box and ash-pit, no castings are necessary for the entire structure, excepting a small frame to protect the charging-hole, which should be situated a little back of the center of the rear hearth, being placed, of course, in the medium longitudinal line of the furnace. It will add also materially to the durability of the fire-box to surround the portions of the same most exposed to pressure or mechanical violence, by light cast plates, held in place by the uprights.

As the portion of the hearth immediately beneath the charging-hole is exposed to excessive wear from the constant precipitation of heavy masses of wet material upon it, an area some six feet square, and in the locality designated, should be constructed of either fire-brick or slag blocks, the latter, from their texture and general physical condition, being peculiarly well suited to the purpose.

By referring to Fig. 23, it can be plainly seen at what stage in construction the various bearing-bars and other iron work must be inserted.

It will be noticed that, instead of adopting the ordinary large ash-pit, entirely open at the rear, according to the invariable English practice, preference is given to a closed ash-pit, to which air is admitted by a door at one or both ends. This effects a great saving in fuel, and brings the process of combustion more perfectly under control. Comparative tests, extending over a considerable period of time, show this saving to amount to about 50 per cent. of the entire fuel consumed, in the case of coal, and about 65 per cent. (in volume) when pine wood is used. The tight ash-pit becomes, of course, a matter of positive necessity where anthracite coal, with a forced blast, is used.

The side and end walls having been carried up to the required height, and the skewback constructed on both sides for their entire length, the carpenters take possession temporarily, usually under the supervision of the head mason, to put in the wooden center on which the arch is to be built. If a second furnace, or indeed any other work, is available for the remaining masons, it is advantageous, though not indispensable, to permit the furnace to stand uncovered for several days, thus allowing the mortar to set, and greatly increasing the strength of the mason work.

Having selected for description that pattern of calciner in which the gradual diminution of the space between arch and hearth, as it recedes from the grate, is due to successive slight elevations of the hearth level, instead of the ordinary downward pitch of the roof, it is evident that the arch throughout its entire extent will be horizontal, while all four inclosing walls are built up to the same height at every point, with the exception of a rectangular flue-opening in the rear wall, 6 by 30 inches.

The construction of the wooden pattern or center is, therefore, extremely simple, requiring only some 20 pieces of 2-inch plank, 16 feet long and 14 inches wide; a lot of 2 by 4 scantling, to form posts about 10 inches in length, four of these being needed to support each plank on edge; and finally, a sufficient amount of 4-inch battens, from one-half to one inch thick, to cover the area of the required roof, when placed about three-quarters of an inch apart. The planks should be perfectly sound, and at least partially seasoned.

By the aid of a long rod, moving upon a pivot at one end, while the free extremity carries a pencil, a segment of a circle corresponding to a rise of 12 inches in the center of the length of 16 feet, should be struck on each plank, and the line followed accurately with a jig-saw.

The segments for that portion of the arch over the bridge and fire-box are shorter, of course, than those belonging to the main hearth, but should be got out in the same manner, and then shortened at each end to the required length.

The scantling should be cut into posts somewhat shorter than necessary to bring the curve on the upper edge of the segments to the proper height for the lower surface of the arch, so that each post may be wedged to an exact bearing with thin slips of wood. It is quite necessary that the weight should be evenly distributed, and each segment, when brought into correct position, is held there by driving a nail through a longitudinal line of battens in the center and at each extremity.

The segments for sloping arches should be still further strengthened by short braces toe-nailed obliquely from the upper edge of one strip to the lower edge of its neighbor, and so on throughout the entire frame.

An omission of this precaution once caused the canting of the segments and consequent destruction of a large, nearly completed arch under the author's charge.

No difficulty will be experienced in removing the wooden pattern in good condition for further use, provided it is supported on small posts as just described; but if long, heavy blocks of timber are used as a foundation for the segments, great labor as well as much injurious sledging must accompany their removal, resulting usually in the complete destruction of both segments and battens. In fact, where this method of support has been practised, it will be found best to burn out the enclosed patterns, after the tie-rods are properly tightened, closing both damper and ash-pit so as to allow only a slow smoldering, and prevent any injurious rise of temperature in the still damp furnace.

Few jobs of mason work require more care and conscientiousness than the laying of a large calciner arch, as, owing to its great width and slight curvature, a very little lack of closeness in its myriad joints would be sufficient to allow it to yield to the enormous pressure brought to bear by its own weight, and become sufficiently compressed to slip down between its side walls. It is quite a simple matter to lay a good solid arch of fire-brick, owing to their great regularity and smoothness, and almost perfect rectangular form; but when red brick are used, which vary so in size and thickness, and are so frequently warped out of all reasonable shape, much care is required.

In ordinary calciners, it is customary to construct that portion of the arch from the fire end of the furnace to a point midway between the first and second working-doors of fire-brick, nine inches in thickness, the brick standing endwise. At this point, or even considerably sooner, when necessary, red brick are substituted, being placed also on end, each brick, after being dipped into a pail of liquid clay mortar, being pressed closely against its neighbor, and finally settled into position with a few light blows of the hammer.

Moderately soft brick are, as a rule, best suited to this purpose, although they must, of course, possess ample solidity to resist the compression to which they are exposed. Hard-burned brick, though stronger, are too irregular and warped to be often used in a large arch, and in any case the brick should be all carefully selected beforehand by the attendant, and assorted in such a manner that each longitudinal row—extending the entire length of the furnace—is composed of brick of about the same thickness.

Another most important precaution is the preservation of the proper angle, as, in order to establish the required curve, each row must incline slightly from the vertical—the lower end of the bricks being in contact, which is not the case with their upper extremities.

The establishment and preservation of the proper curvature are facilitated by the occasional interpolation of a longitudinal row of wedge-shaped or key-brick, technically called "bullheads." These are usually only obtainable made from fire-clay, but are almost indispensable for the center row when the final keying of the arch is effected. Otherwise, the entire row of key-brick must be cut from common brick, an arduous and imperfect task.

The keying is a matter of some delicacy, and should be performed by a single workman, who should select or cut his keys of such thickness as to produce a uniform moderate pressure throughout the entire distance, no more force being exerted to drive the key into place than can be easily effected by a light mason's hammer, using an intervening block of wood to prevent the destruction of the brick.

While the masons are thus employed, the blacksmith and his helper should have completed the buckstaves and tie-rods from measurements furnished by the foreman mason as the work progresses, it being in such cases easier to suit the length of the tie-rods to the completed mason work than to pursue the opposite course.

As soon as the arch is completed, the head mason and blacksmith should proceed to the ironing of the furnace, which, with the assistance of two laborers, should be completed in a single day.

The most convenient and easily obtained buckstaves in many cases are old iron rails of full size, say, 80 pounds to the yard. Properly shaped I-beams, of corresponding strength, are about 15 per cent. lighter. The tie-rods may consist of one inch round-iron for the bottom rod, and one inch and a quarter iron for the upper rods. The lower rods are already long in place, and through each of their loops should now be slipped one of the upright buckstaves, cut to the proper length, and temporarily wedged into the loop to keep it perpendicular.

The upper tie-rods may be made the same as the lower, with a loop at each end—the necessary tightening being effected by flat iron wedges; or they may have a threaded extremity at one end passing through a corresponding hole in the buckstaff, and fitted with a strong nut; or, best of all, a small ring is formed at one end of the tie-rod, through which slips a U-shaped piece of round iron, which fits against the buckstaff, on the other side of which a piece of flat iron, pierced with two holes for the free ends of the U is held, these ends being threaded; a nut for each of the ends completes the apparatus, and presses the piece of flat iron tight against the upright. This is a simple and highly satisfactory device, and avoids the disagreeable process of wedging in the one case, or of punching a large hole through a narrow rail in the other. The strain is distributed over two bolts and nuts, and can be instantaneously increased or diminished; nor will the nuts rust solid into place, provided they are saturated with oil annually, and slightly turned, to free them.

Whatever method of tightening the tie-rods may be selected, the process of ironing or anchoring should begin with the first tie-rod on the *main body* of the furnace, nearest the fire end, and proceed systematically toward the rear, thence returning to the shorter transverse rods that support the arch over the grate, and terminating with the long longitudinal rods, which, for convenience of handling, should be in three lengths, connected with hooks and eyes. Up to this time, no great strain should be put upon the rods, everything being merely brought to a solid bearing; but after all are in place, and the buckstaves evened both vertically and laterally, the rods may be drawn to the desired tension, the skew-back being still further supported by a bar of one by four-inch flat iron, or, better, an iron or steel rail, let in flush with the brick-work.

This is largely a matter of experience, and being of vital importance should receive the most careful attention on the part of the builder, as too lax a condition of the rods may permit the entire falling in of the arch, while the contrary fault may cause a positive buckling and elevation of the same, accompanied with a general cracking and distortion of the lateral walls. The latter accident, in a moderate degree, is much more likely to occur than the former, owing to the natural tendency to overdo a measure essential to safety, and yet not exactly defined.

The lateral rods should be tightened until they begin, when struck near the center with a hammer, to vibrate rapidly, and to be but little depressed when stepped upon. (It is almost needless to say that none of the upper rods should touch the arch.) A simultaneous examination of the brick-work forming the upper portion of the side walls should also be made, as it is here that the effect of the curving of the buckstaff from too great tension, and consequent pressure against the mason-work, is first visible.

The extreme limit of tension is reached when the first signs of this appear, as nothing can be gained by bending the uprights, and if the latter are sufficiently strong and applied in the numbers shown in the illustration, the arch may be considered perfectly supported. All the rods should be tightened to about the same extent, although it must be remembered that the great length of the longitudinal rods may prove deceptive in estimating their tension, it being impossible to tighten them to such a degree as the shorter lateral ones.

A single additional precaution is recommended, though seldom practised by builders. This consists in breaking up a few thin roofing slates into fragments a couple of inches in length, and driving these with moderate force into whatever crevices may still be found in the surface of the arch.

Some twenty or thirty pails of liquid mud are now poured over the arch, and the process repeated as it dries until every crack and crevice is filled, and the roof rendered completely solid and air-tight.

The wooden center on which the arch was built should now be removed by first knocking away the little posts that support it, using a light stick of timber as a battering-ram, and proceeding from one side door to the next until every stick and batten are removed. They should be stored for future use. Any indications of settling on the part of the arch must be immediately counteracted by tightening the tie-rods; but when the precautions enumerated have been carefully observed, this can never occur.

The length of time the completed furnace may now stand untouched with advantage to the mason-work is only limited by the requirements of the business, which almost invariably demand its being put in commission at the earliest possible moment. Under such circumstances a smoldering fire of large logs, knots, or any slow-burning waste material, should at first be kindled on the floor of the ash-pit, the grate-bars not being put in place until the masonry surrounding the fireplace is partially dried.

In twelve or eighteen hours the fire is elevated to its proper place, and with a nearly closed ash-pit door and partially lowered damper, the process of drying proceeds gently and without that violent generation of steam and vapor that is sure to be accompanied by extensive fissuring of the brick-work and permanent weakening of the entire structure.

A most careful and repeated examination of the condition of tie-rods and buckstaves should be made every few hours from the first kindling of the fire until the furnace has attained its full heat, and may be supposed to have expanded to its utmost limits, although it may be a month or more before all evidences of movement cease. The first indication of this process will be seen in the neighborhood of the bridge and fireplace, where the highest temperature prevails. A bending of the buckstaves, combined with a pressing in of the skewback line and an increased tension of the cross-rods, are warnings that may soon be followed by either a complete giving way of some portion of the iron-work, or more frequently by a bodily upheaval of the arch and general fissuring of the brick-work unless relieved by diminishing the strain to a corresponding degree. This process of loosening must be extended to the entire iron-work of the furnace, and continued as long as necessary, the tension being again increased if the furnace is ever allowed to cool down to any considerable degree—an operation more destructive to it than many months of ordinary wear.

While the apparatus is thus gradually being brought into proper heat, the sheet-iron hopper should be suspended from timbers resting upon the trussed beams of the building. It should be strongly constructed and well braced, and provided with a stout lever, easily accessible to the operator when standing upon the floor of the building. A track running transversely to the row of calcining-furnaces, and parallel with the longitudinal axis of the building, renders these hoppers easily accessible to the car in which each weighed charge of ore is brought. The car is provided with a dumping arrangement, so that it easily and completely empties itself into the furnace hopper. The laborer who weighs and transports the charges can supply six furnaces, provided everything is arranged as herein described, or in a similarly judicious manner.

The outfit of tools may now also be prepared, and should consist, for each four-hearth calciner, of 6 rabbles, 4 inches by 10 inches and 12 to 14 feet long; 6 paddles, 8 inches by 12 inches and 14 feet long; 4 door-hooks, to handle the sheet-iron working-door; 1 longhooked and pointed iron poker for wood, or an ordinary coal poker, if coal is used; 2 ordinary long-handled, square-pointed shovels; 1 scoop-shovel (for coal).

The iron rollers, usually employed as rests for the long tools at each working-door, soon lose their shape and cease to revolve. It is better, therefore, to provide merely a smooth iron bar, which, if kept well soaped, renders the handling of the tools as easy as any of the more expensive devices.

When available, a free-burning semi-bituminous coal forms the most economical fuel for calcining purposes, but should always be burned upon a comparatively shallow grate, instead of using the deep clinker bed, so suitable to the smelting process. At the comparatively low temperature suited to calcination, the generated gas does not burn perfectly, and a great waste of fuel occurs. Coal should be fed at short intervals—from 30 to 45 minutes—in quantities seldom exceeding 50 pounds. When wood is cheap, nothing can excel it as a fuel for calcining purposes, its long, hot, non-reducing flame being peculiarly suited to the requirements of the process. About one and two-thirds cords of hard, or two cords of soft wood are commonly considered equal to 2,240 pounds of fair bituminous coal.

CONSTRUCTION OF CALCINER STACKS.

The most important feature of a chimney is its foundation; but it is at this very point that a great saving over ordinary practice may be effected without lessening the stability of the superstructure.

A mere increase in depth below the loose soil forming the surface of the ground does not add in the slightest to the value of the foundation, after a proper material for the same has once been reached; and as this occurs in the greater number of cases within three or four feet of the surface, the frequent practice of additional excavation for the apparent purpose of merely gaining depth is money thrown away.

After removing the loose surface soil, and penetrating below any danger of frost, in the greater number of cases no advantage would be gained by excavating to a depth of 50 feet, unless solid bed-rock were reached.

Any kind of gravel, hard-pan, or even soft loam or sand, if homogeneous, will answer the purpose perfectly, it being understood that reference is here made to an ordinary calciner or smelter stack not exceeding 80 feet in height.

In the case of a yielding sand bottom, and especially if the line of division between two strata of varying quality happens to cross the excavation, it is well to form a solid floor to the pit by putting in a double layer of 3-inch plank, nailed crosswise. But in all ordinary cases the hole should be simply filled with broken stone, about the size of ordinary road metal. This material, when well rammed into place and thoroughly grouted, by pouring in a sufficient quantity of mortar composed of one part each of lime and cement, and three of sand, makes a foundation infinitely superior to one formed of a few large stones, the slightest settling of any one of which will throw the chimney out of perpendicular.

The excavation should be at least three feet larger in every direction than the base of the chimney, and the stone-work of the latter, laid in lime and cement, may cease some three feet below the surface, at which point the brick-work usually begins.

If a smelting-furnace is in operation in the immediate vicinity, nothing can be more satisfactory or economical than the following plan, pursued by the author on several occasions:

An excavation being made of the usual size, the molten slag from the smelting-furnace is wheeled to the spot in the usual movable slag-pots, and poured at once into the hole, which, when filled to the proper height with the fused rock, and leveled by means of little clay dams along the edges, so as to present a smooth surface for the masons to begin on, will contain a solid block of lava, weighing many tons, and as immovable as a ledge of rock.

In constructing a stack we have to determine the size of flue desired, and intimately connected with the same is the degree of *batter*, or taper, which shall be given to the structure.

The object of this batter is twofold: 1. For appearance. 2. For the sake of strength. The first reason may be entirely neglected in metallurgical architecture, and experience has shown that, within the limit of height mentioned, a batter of one-eighth of an inch to the foot is ample. Nor need the taper be begun until the stack rises above the roof, as that portion of the structure within the building is amply protected from the force of the wind.

By thus decreasing the amount of taper, we greatly increase the capacity of the stack, as experience shows that a contraction of the flue in its upper portion is accompanied with a corresponding diminution of draught, while a positive enlargement of the same toward the top has a most beneficial influence. This latter point is gained by lessening the thickness of the chimney walls as they grow higher, while the outside taper remains constant.

All calculations and formulæ regarding the necessary size of any flue for a given duty have been found so greatly modified by circumstances—such as variations of internal and external temperature; humidity of atmosphere and state of barometer; change of winds, etc.—that it is found safest to rely upon experience and analogy; and after beginning with a much larger flue for safety, the author has finally found a stack 42 inches square inside, at its narrowest part, and 65 feet high, to possess ample capacity for two large calcining-furnaces such as just described. It is proper to add that a much smaller stack will produce the draught usually considered as quite sufficient for the calcining process; but long-continued experiment has shown such extraordinarily favorable results, as regards both capacity and perfection of roast, to arise from greatly increasing the ordinary calciner draught, that a sharp and powerful draught appears as essential to a calciner as to a smelting-furnace.

For this reason, also, no more than two furnaces should be led into a common low stack, it being almost impossible properly to equalize the admission of air to each calciner, and to produce that sharp and vigorous draught so essential to rapid oxidation, and especially to the conveyance of the sheet of flame and heated gases over the whole length of a 4-hearth calcining-furnace. The interposition of dust-chambers, or preferably of large flues, filled with parallel rows of sheet-iron, according to the method found so efficient and economical at Ems, is of course necessary, and should be present in any case.* Limited experiments conducted by the author fully satisfy him of the great benefit to be derived from the adoption of this economical and efficient method of condensation.

The size of chimney mentioned—42 inches—will answer for all elevations up to 5,000 feet above sea-level. For each 1,000 feet additional height, these figures should be increased one inch.

For a calciner chimney of this size and 65 feet in height, the walls at the base should be 17 inches thick, the length of two red brick, no fire-brick being needed, as the gases are sufficiently cooled by their passage through the long furnace and flue. This thickness is maintained for a height of 25 feet from the ground,

* See description of Ems method of condensation, by Professor Egleston, in *Transactions of the American Institute of Mining Engineers*, XI., 379.

which brings it somewhat above the roof of the building. At this point, the external batter of one-eighth of an inch to the foot is begun, and an internal set-off of 4 inches is taken; thus decreasing the thickness of the walls to 13 inches, and enlarging the flue to 50 inches.

This constant taper is maintained by the employment of an ordinary beveled plump-bob, which obviates any trouble or calculation. This condition of affairs is continued for another 25 feet, during which distance the flue is contracted to a size of about 44 inches, when another internal 4-inch set-off is taken, increasing the same to 52 inches, while the walls are diminished to 8 inches.

This, being continued for 15 feet, gives the full height of 65 feet, the flue at the top being still 48 inches square, or 6 inches larger than at the base. No ornamental finish at the top should ever be allowed, the stack either being surmounted by a light casting to hold the brick in place, or left without this protection, the iron braces being usually sufficient to prevent the loosening of the upper rows of brick-work. An ornamental cap is simply a source of annoyance and danger, and should never be permitted in a stack devoted to the passage of sulphurous vapors.

A chimney of this size is best built from the outside, a scaffold being erected by placing eight stout poles about the base of the proposed structure, nailing crosspieces at the proper height for the plank staging, and thoroughly bracing the uprights by boards nailed diagonally from one to the other.

The uprights may be lengthened out almost indefinitely by careful splicing, and as the stack grows higher, new crosspieces are spiked every five feet, and men and material thus maintained at the desired elevation. A rope and bucket, with a single wooden block fastened to the railing of the staging, and manipulated principally from the ground level, form the most economical means of elevating the requisite material, while a single laborer above is able to furnish four masons with brick and mortar, most of the work being done from below. It is best to employ four masons, so that one can work on each wall of the stack, and their position should be changed twice daily, in order to equalize any differences in the amount of mortar used, etc.

Like all other mason-work that is to be exposed to heated sulphurous gases, the interior portion of the stack must be laid in clay mortar (ordinary sticky mud); while the remainder of the structure should be laid in lime mortar, on account of its superior tenacity. To prevent the penetration of the vapors into the porous brick, the interior of the flue should be thoroughly plastered with clay throughout its entire extent.

While the durability of a chimney of this description is largely dependent upon its being ironed, it is still more dependent upon its not being ironed too stiffly. A stack with corners thoroughly inclosed in stiff angle iron, tightly held together with frequent braces, may fissure and give out in a few years, while a similarly built chimney containing a few light irons, merely to hold the brick-work in place, will last twenty years or more.

This is the result of personal experience, confirmed by the observations of most other constructing engineers, and is especially the case in countries where high winds and violent fluctuations of temperature are prevalent.

Eight uprights of $\frac{3}{8}$-inch by $\frac{3}{4}$-inch iron, each upright being placed about 4 inches from each corner of the stack, and passing through rectangular openings cut in one-half by 2-inch flat iron, which latter pieces are laid in the brick-work from 30 to 36 inches apart, are amply sufficient for the purpose. The holes must be so punched that the uprights can be wedged tightly against the brick-work, which is thus held in place even after the mortar has long succumbed to the combined influence of the roast gases and the elements. As a striking example of the accuracy of the above remarks, the reverberatory smelter stacks of the Detroit Smelting Company's copper-refining furnaces at Lake Superior may be mentioned, where, on building a strongly ironed stack, they found it fissure and become unsound in a very short time; whereas their ordinary stacks, anchored only by means of occasional straps of flat iron built into the chimney walls and bent over at each end, have stood for fifteen years or more without showing crack or imperfection.

A row of headers should be introduced at about every eighth course, and the lower portion of the stack into which the two calciner flues enter on opposite sides should be divided by a 4-inch partition wall into two equal compartments. This wall, extending some five feet above the entrance flues, serves to bend each current in an upward direction, and thus prevent the whirl and disturbance of draught resulting from the meeting of two opposing currents.

The following interesting observation has been communicated by Messrs. Cooper and Patch, superintendent and chemist of the Detroit Refining Works:

In most reverberatory furnaces, the flue enters the stack at some distance above its base, and consequently there is a cavity inclosed by the chimney walls, of greater or less depth below the embouchure of the flue. When this apparently useless cavity has become filled up from the falling in of the stack lining, drippings from the molten brick, or other causes, the draught at once suffers and the capacity of the furnace is greatly diminished.

Whether this phenomenon arises from the loss of the elastic air-cushion that is normally present, or whether there is some other reason, the fact remains, and although the observations have been confined mostly to smelting-furnaces, it is probable that a calcining-furnace may be affected in a similar manner, and therefore in all cases where a horizontal or inclined flue enters a stack, it should be so constructed as to leave an open space of from 4 to 6 feet below it. This need not communicate with the outside air in any way, except for the purpose of cleaning the stack or entering it for repairs.

It is well to provide every high stack with a good lightning rod, properly fastened and insulated.

The building that covers any considerable number of calcining-furnaces is necessarily of great extent, and should, if possible, be built of very light and, at the same time, fireproof materials.

Scarcely anything fills these requirements so thoroughly as a medium grade of corrugated iron. This, if well fastened down, and painted every three or four years, will be found the most economical and satisfactory material for both sides and roof that is yet known. If the number of furnaces under a single roof exceeds two, they should be placed at right angles to the greatest length of the building, a space of only three feet being left between the rear end of the furnace and the corresponding side of the building, while between the fire-box and the lower side of the building there should be ample room for a driveway for the conveyance of fuel, as well as for a railroad parallel to the same and close to the wall, over which the calcined ore may conveniently be dumped into a paved and roofed inclosure on a level as low as the circumstances of the case permit. The 16-foot calciners should be separated by spaces of at least fourteen feet.

As the main building for these long calcining-furnaces must be from 80 to 90 feet in width, it is often the practice to support the cross-beams on posts that, if properly placed close to the furnace and midway between the working openings, need not interfere with the long tools in use. But there is no difficulty in constructing trusses to support a roof of this size without the aid of posts, nor need the expense be much greater. The principal difficulty is encountered in raising these immensely long and heavy "bents;" but this may be entirely obviated by constructing a series of cheap scaffoldings, and putting them together piece by piece, instead of attempting to raise the entire "bent" bodily. The ridge of the roof should be surmounted by a continuous ventilator throughout its entire extent. The details of this work may be intrusted to any experienced carpenter.

COST OF CONSTRUCTION OF CALCINING FURNACE.

The following estimates of cost are taken from notes that cover the construction of a considerable number of large calcining-furnaces, and being given without alteration or omissions, excepting the necessary reduction to our assumed standard of costs, should furnish reliable figures on which to base future plans:

COST OF ONE FOUR-HEARTH CALCINER.

Excavation—45 days at $1.50	$67.50	
Removal of material excavated	35.00	
Superintendence and miscellaneous	24.00	$126.50
Foundation walls—1,840 cubic feet.		
2,000 slag-brick at 2 cents	40.00	
20 days stone-mason and helpers	120.00	
Materials for mortar	28.00	
Labor on same and utensils	16.00	
Miscellaneous labor	12.00	
Superintendence	15.00	$231.00
Brick-work on furnace proper.		
2,420 cubic feet, say 50,000 red brick at $8	400.00	
7,500 fire-brick at $40	300.00	
Lime and sand	187.00	
4 tons fire-clay at $8	32.00	
8 tons brick-clay at $1.50	12.00	
32 loads sand at $1.50	48.00	
112 days' brick-masons' labor at $4	448.00	
112 days' ordinary labor at $1.50	168.00	
8 days' carpenters' labor at $3	9.00	
Miscellaneous labor	35.00	
8 days, blacksmith and helper	40.00	
Materials consumed by same	8.00	
Superintendence	112.00	$1,749.00

Carried forward		$1,749.00
Iron work.		
66 buckstaves (old rails), 6¼ feet long, 80 pounds, at 1⅝ cents per pound	85.80	
Tie-rods and loops, 2,056 feet, 1¼-inch round iron — 8,327 pounds, at 2 cents	166.54	
Flat-iron for skewback, grates, etc. — 2,064 pounds, at 2 cents	41.28	
16 cast frames and doors, at 156 pounds each — 2,496 pounds, at 2½ cents	62.40	
Fire-doors and other small castings	16.50	$372.52
Nuts and bolts		6.25
Short flue with damper, and one-half cost of stack		364.00
Grading and miscellaneous		47.50
Tracks for feed and discharge of ore		62.40
Set tools, complete, as per former schedule, 1,250 pounds, at 2 cents		25.00
Labor on same		18.00
One iron-ore car (list price)		85.00
Grand total		$3,087.17

The repairs on a thoroughly built calciner should be nothing for the first three years; for the succeeding seven years they will average 3 per cent. per annum on its first cost, while from its tenth to its fifteenth year, 5 per cent. per annum will probably be expended in renewing the hearth and roof once and patching the furnace in various places.

After fifteen years of constant usage, it is cheaper to build a new furnace than to keep the old one in repair; but few metallurgical enterprises in this country require to provide for a period longer than the above.

COST OF CALCINING IN HAND REVERBERATORIES.*

In roasting a heavy pyritous ore for subsequent reverberatory smelting, as at Butte, Montana, where concentrates with 40 per cent. sulphur require to be roasted down to 7 or 8 per cent. sulphur, a large calciner, properly and energetically managed, will put through 13 tons of ore per 24 hours with a consumption of 2

* As most of the costs of calcining in this work are based on Montana or Colorado prices, the estimate of the cost of calcining, as given in earlier editions, has been changed to correspond with the other calcining estimates. It has also become customary to burn more coal in calciners than formerly, and drive them harder, thus increasing the capacity of the furnace and often the cost per ton of material roasted.

tons of slack coal and with the services of four men working 12-hour shifts in two gangs.

The following estimate shows the minimum expenses:

EXPENSES AT CALCINER PER 24 HOURS.

Two tons slack coal at $3.50	$7.00
Four furnace-men at $3.50	14.00
One-fourth weighman at $3.00	.75
Repairs, lights, and miscellaneous	.60
Proportion of foreman	.50
Interest on $4,000 at 6 per cent. per annum	.66
Total	$23.51

Or about $1.81 per ton of raw ore.

(*b*) MUFFLE CALCINERS.

The variety of reverberatory calciner known as the *muffle furnace* is now seldom used by the copper smelter, as, except for purposes of acid manufacture, it possesses few advantages above the ordinary hearth variety, and in case this branch of metallurgy is also practised, some of the newer forms of automatic furnaces have displaced the muffle. The high cost of construction and greater consumption of fuel are also adverse to its employment, and although, from its gentle and regular heat, it possesses decided advantages in the treatment of easily fusible substances, it is rather suited to the calcination of matter containing much lead, or of pyrites with salt, as in the Henderson process, none of which operations come within the scope of this treatise.

An easily fusible ore can be very efficiently protected from the fierce heat of the first hearth of an ordinary calciner by the construction of a 4-inch curtain arch, covering one-third or more of its surface from the fire-bridge onward, though such a precaution is seldom necessary, excepting in the case of matte calcination, which requires but slight modifications of the roasting process as applied to ordinary sulphide ores.

REVOLVING CYLINDERS.

These also are extensively and advantageously used for the chloridizing of silver ores, having a considerable capacity, and effecting a thorough chloridization at a very moderate cost. They consist essentially of a horizontal or inclined brick-lined iron cylinder, revolved slowly by gearing, and having a fireplace at one end —or at both ends, used alternately.

(*a*) CYLINDERS WITH CONTINUOUS DISCHARGE.

The cylinder with continuous discharge that has been most largely used for the oxidizing roasting of copper ores or products is the *White-Howell* and its imitations.

This is a slightly inclined cylinder of small diameter in proportion to its length, is lined with brick, and is cradled between supporting rollers, being slowly revolved by means of pinion and spur-gear. Four longitudinal ridges of brick-work project slightly from the inner lining at intervals of 90 degrees, and lift the ore until it falls back through the flame in a thin stream, and is continuously discharged at the fire-box end of the cylinder.

The amount of ore treated is determined by the speed at which the furnace is revolved, and the angle of inclination at which it is set.

It is largely used for the chloridizing roasting of silver ores, but, in spite of its many seeming good points, has never been very popular among copper men.

At the works of The Cape Copper Company, Limited, at Briton Ferry, and The Messrs. Elliott's Company's Works at Pembrey, South Wales, I saw several cylinders of the above general pattern running on 76 per cent. white metal. The cylinders were 7 feet in diameter and 60 feet long, having an inclination of 5½ inches. They made 8 revolutions per hour, and calcined about 22,000 pounds of the metal per 24 hours down to 1 per cent. to 3 per cent. sulphur. The consumption of coal was 2,000 pounds in the fire-box, besides the power.

The matte, crushed through a screen with three meshes to the linear inch, was conveyed into an iron hopper at the cold end of the furnace, whence it flowed by gravity into the cylinder through a two and a quarter by half-inch slot in the floor of the hopper. The cylinder required one-fourth of a laborer's time to fire and remove ashes, and a small lad to watch the feeding, oil the machinery, etc.

The cost per 2,000-pound ton of matte, taken from the results of several years' running, and allowing for repairs and interest on investment, was reported to me as 33 cents. This did not include the handling and re-roasting of the flue-dust. These costs, naturally, were based on Swansea prices for coal and labor, but they are extremely low, and are interesting as showing the ease with which certain sorts of matte can be calcined in continuous cylinders.

The continuous discharge, muffle cylinder-calciner of James Douglas contains a heavy, central tile-flue, supported by four slotted tile partitions. The products of combustion are thus carried direct into the chimney, without ever mingling with the roast gases.

Mr. Douglas invented this apparatus for the calcination of pyrites fines, in order to obtain pure and concentrated sulphurous acid fumes for the Hunt and Douglas process of wet copper extraction, but it has proved a rapid and efficient calcining furnace as well.

After the heavy interior mass of brick-work has become once thoroughly heated, the cylinder will do good work without the aid of carbonaceous fuel. Its capacity is largely influenced by the amount of air admitted to the roasting chamber; and, as the primary mission of this apparatus is to furnish a supply of concentrated sulphurous acid gas, its actual roasting capacity has always been seriously handicapped. Its capacity is 6 to 12 tons of pyrites fines per 24 hours, roasted down to about 3 per cent. sulphur.

(*b*) CYLINDERS WITH INTERMITTENT DISCHARGE.

At the present time, the modified and improved *Brückner's cylinder* stands pre-eminent as the most satisfactory and economical of all revolving cylinders for pulverized ore.*

The large cylinders, as now made, are 8 feet 6 inches in diameter by 18 feet 6 inches long.

They are lined with one thickness of good red brick, though doubtless, where fire-brick are cheap, it will pay to use them, as they withstand the mechanical wear and tear much longer than red brick; owing to the care bestowed upon their manufacture, they are much more regular in shape, thus forming a tighter and more perfect circle inside the iron shell, the strength of which can be still more increased by having the brick molded to order to suit the inner circle of the cylinder. Again, they are much more durable when exposed to dampness than are red brick, which are

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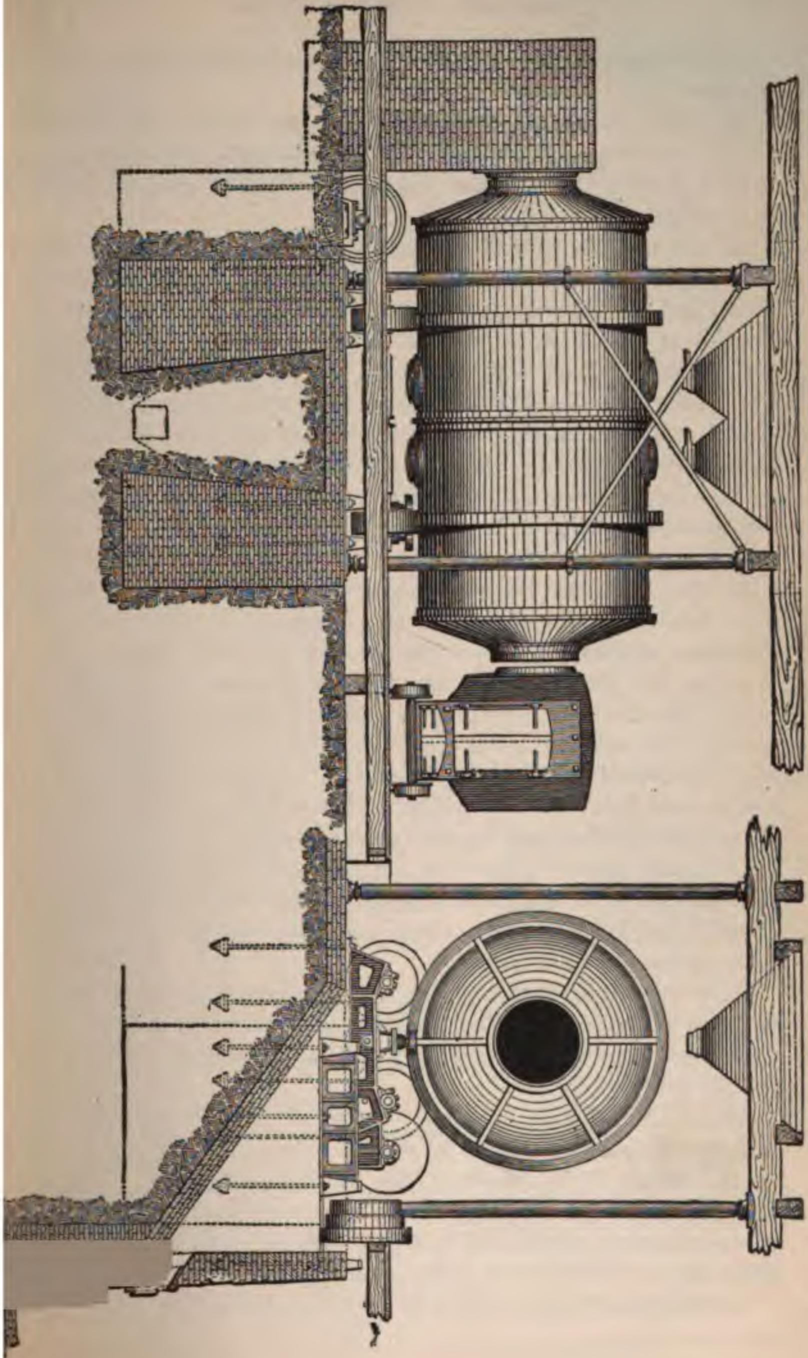
* In describing this furnace, it would be unjust not to mention Messrs. Fraser & Chalmers, of Chicago, whose energy in introducing it, and in going to great trouble to modify and improve it, and adapt it to the desulphurization of copper ores and concentrates, has earned the gratitude of all copper-smelters. To Mr. W. R. Eckart, the profession is indebted for the detailed drawings of the cylinders now giving such satisfaction at the Anaconda Works, in Montana.



quickly destroyed if dripping concentrates are fed into the red-hot furnace.

But even where red brick are used, the lining lasts about 18 months when properly put in, and as this is the principal cost of repairs during the first few years, it is evident that it must be very small.

As will be seen in the accompanying perspective sketch of the furnace, it has a double-snouted feed-hopper, with two feeding-holes, and two others opposite them, halfway around the circumference of the shell, so that it can be discharged without much loss of time. It is best, of course, where possible, to discharge the roasted ore directly into the reverberatory smelting furnaces, if such are used, or into an adjacent vault, where the heat will not be rapidly dissipated. But in works where the calcined ore must be first cooled down before going to the smelter, the cooling arrangements must be of large capacity to handle the heavy charge of ore employed.

In the improved cylinders, the fire-box is really a car, running on a track at right angles to the longitudinal direction of the cylinders, and having a short flue in one side that comes exactly opposite the throat of the furnace. In this way, the fire-box can be run opposite a cylinder, which contains a fresh charge, and fired on until the sulphur is fairly kindled. Then the movable fire-box may be wheeled along to a neighboring cylinder, and the first one left to complete the combustion of the sulphur with a free access of air, and undisturbed by the reducing gases that pass through an ordinary grate. After the combustion of the sulphur, it is necessary for a perfect roast to again connect the fire-box with the cylinder and supply a little extraneous heat to complete the decomposition of the sulphates.

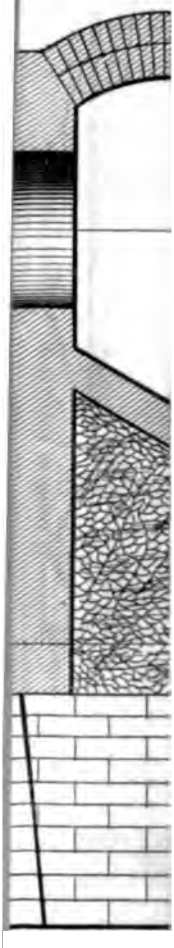
It is estimated that two horse-power are required to drive a charged cylinder at average speed. The size and weight of the ore-charge varies greatly with its quality, percentage of sulphur, specific gravity, etc.

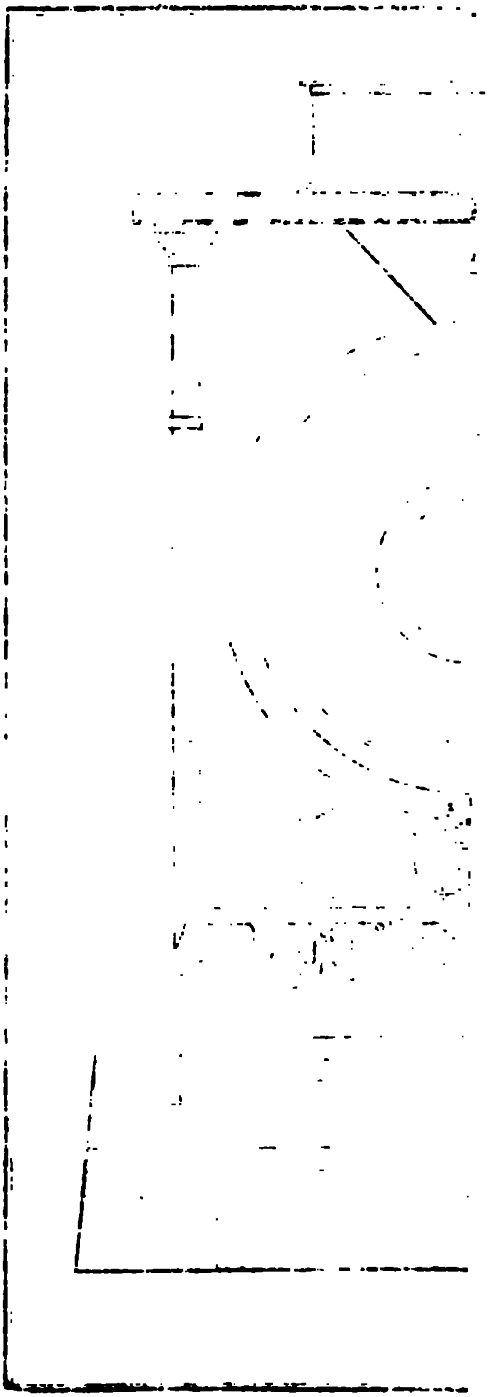
These results were obtained at the works of The Anaconda Mining Company of Montana, where 156 of the cylinders, 8 feet by 18 feet, are in operation. The following results comprise the work of four weeks (28 days). The ore calcined consists mainly of concentrates containing 36.4 per cent. sulphur and 16 per cent. silica and is roasted down to 8 per cent. sulphur.

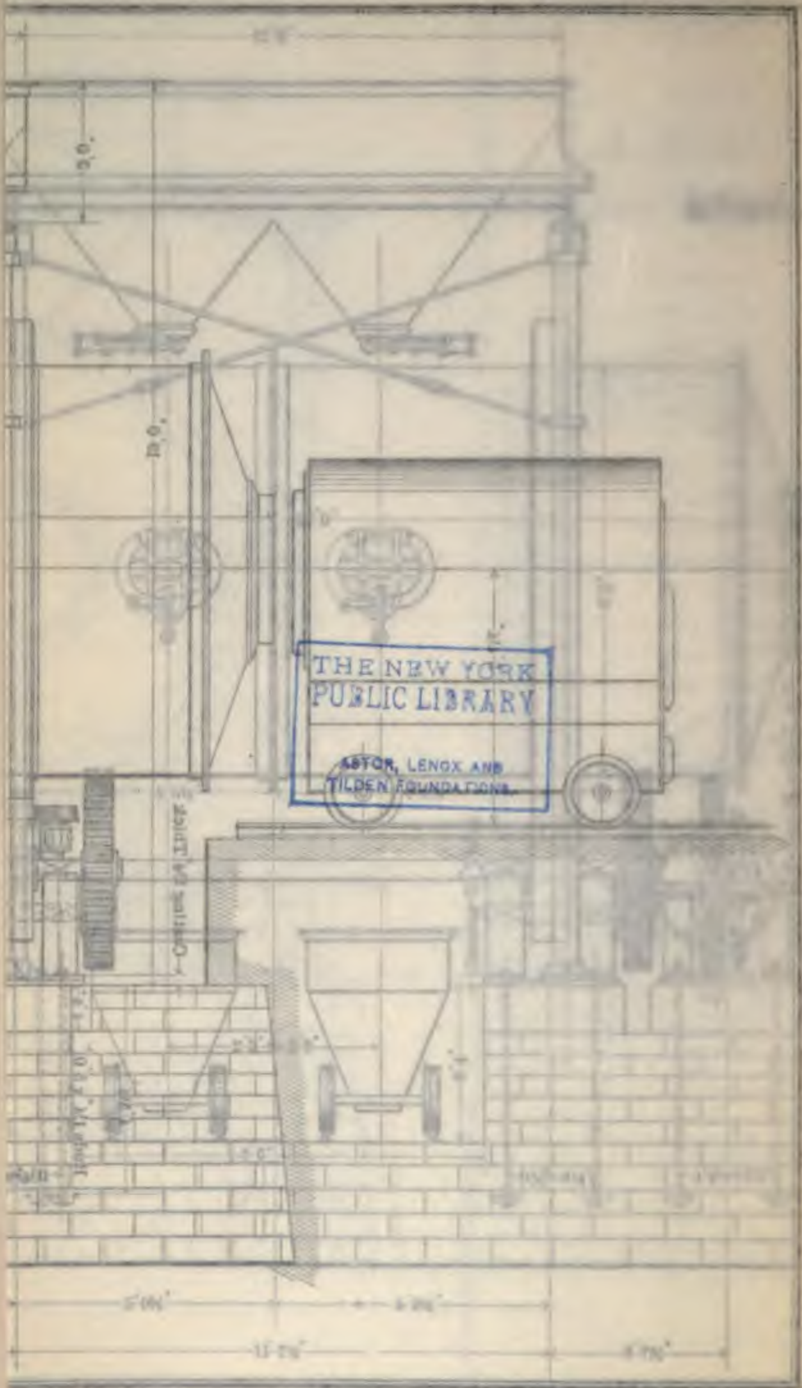
In 28 days a cylinder treats 341 tons of dry ore or 12.186 tons

PLATE

KNER'S







per day. It uses 2.95 cords of wood (378 cubic feet) per day, costing $10.26, or 84½ cents per ton of ore. This wood could be unprofitably replaced by 2.63 tons inferior coal, with 20 per cent. ash; or by 1.625 tons of better coal, with 10 per cent. ash. One laborer, at $3 per shift of 12 hours, attended three furnaces, costing per day per furnace, $2, or 16.4 cents per ton of ore.

This makes $1.01 per day, to which must be added the expense of power, repairs (small), recrushing and recalcining lumps and rehandling flue-dust (large), oil, lights, foreman, and interest on plant, $6,000. This brings the total cost to about $1.40 per ton of ore.

By the courtesy of the Chicago Iron Works, I am enabled to present some excellent drawings, showing the details of the Brückner cylinders manufactured by them, and which are doing very satisfactory work in Montana and elsewhere. (See Plate I.).

Direct statements from those who are using them show that my estimate of a saving of 30 to 40 per cent. of the costs of calcining by using these large cylinders in lieu of hand-calcining furnaces is by no means excessive, and in some instances does not represent the full amount saved.

CHAPTER VIII.

AUTOMATIC REVERBERATORY CALCINERS.

AUTOMATIC hearth-furnaces seem to offer peculiar advantages as regards capacity in proportion to first cost, and ease of management. They are also used for the roasting of leady mattes and other material that is inclined to sinter. They seem peculiarly suited to roasting pyritic gold ores and concentrates, previous to their treatment by chlorination; nor can I see why they could not be changed into muffle-furnaces, that, considering the space, labor, and plant saved, would roast pyrites for sulphuric acid manufacture more economically than any of the burners at present in use.

The most prominent furnaces of this description now before the public are:

The O'Harra furnace with certain modifications by Allen and by Brown.

The Pearce turret furnace.

The improved Spence (Keller-Gaylord-Cole) furnace.

The Brown horseshoe furnace.

The Spence automatic desulphurizer.

To these may be added two furnaces that are solely engaged in roasting zinc-blende ores, but that possess features of interest to the copper metallurgist, viz.:

The Matthiessen & Hegeler Zinc Company's furnace, working at La Salle, Illinois.

Blake's revolving hearth calciner, at Shullsburg, Wisconsin.

The O'Harra furnace consisted originally of two long hearths, one above the other, through which plows were continually dragged by means of a chain which obtained its motion from grooved pulleys over which it ran. The chain and plows were, and still are, cooled by running for some distance outside of the furnace on their way from the lower to the upper hearth. The hearths contained a continuous, longitudinal groove, not for the chain to run in, as is stated by Schnabel and certain other writers, but for the protection of the chain in case of shut-downs, whose probable frequency and extent were evidently very apparent to the inventor. During such delays the tension was relaxed, and the chain was supposed to subside into the groove. The hearths were heated by a sufficient number of external fireplaces along their sides.

While the capacity of this furnace was large, and the roasting satisfactory, the repairs and delays were excessive. The chain and plows riding on the hearth, constantly gave way and wore out. The plows tore up the hearth and dragged it to the front, and the life of a furnace scarcely reached eighteen months.

Allen effected a radical improvement by laying iron tracks through the hearths, and mounting the chain and plows on wheeled carriages.

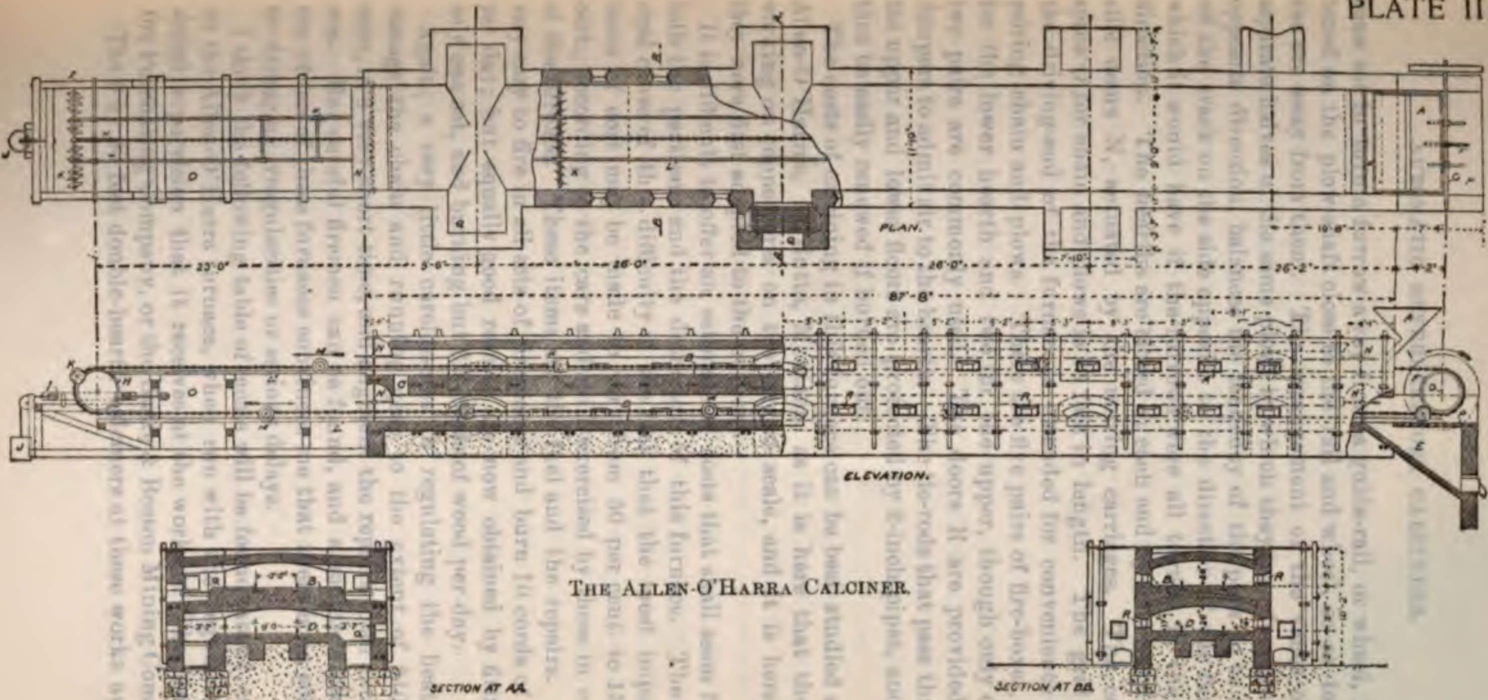
Brown's modification consisted in partitioning off little corridors on either side of the hearths, in which the tracks were laid, and through which carriages ran, supporting the chain, and especially the arm to which the plows were attached, one end of this arm being fastened to the carriage, while the other extremity projected through the partition wall of the corridor into the furnace. It is evident, therefore, that Brown had to use two chains, and two sets of rabbles, their arms nearly meeting at the center line of the hearth. It is also evident that there had to be a continuous slot in the partition wall, to permit the travel of the rabble-arm.

This modification has not been so entirely successful as its ingenuity would seem to deserve. The main difficulty has been the tendency of the partition-tiles or castings to sag or loosen, and obstruct the continuous slot, through which the rabble-arm projects into the hearth. This trouble has been remedied by Brown, and independently by the Argo metallurgists, but a mere partial partitioning off of the hearth does not seem to be a sufficiently perfect means of protecting the tracks, carriage and chain from the heat. Those who are running O'Harra furnaces claim that the chain, track, etc., might about as well be in the hearth proper, as it was before Brown's modification, and most of the O'Harra furnaces are run in this manner. Brown is entitled to great credit, however, for showing us the use of a continuous slot traversed by a rigid rabble-arm.

The Allen-O'Harra calciners at Butte have two hearths, each 9 by 90 feet, traversed by six plows, making a complete circuit in 3¾ minutes. They roast highly pyritic concentrates containing about 19 per cent. copper, 18 per cent. silica, and 40 per cent. sulphur, down to 8 per cent. or 9 per cent. sulphur. A considerable proportion of the concentrates are coarse, and all are wet. They lose 26 per cent. of their dry weight by calcination.

The weakest point of even the improved O'Harra furnace is its heavy repair bill. This is, to a considerable extent, unavoidable in a furnace where the track, carriages, and chain are all exposed to the flame and to the red-hot sulphides, and where their existence is entirely dependent upon the judgment and care of the firemen. But both Allen and Bellinger of Butte, and Fraser & Chalmers of Chicago, have introduced modifications that considerably lessen the cost of repairs. The wear on the carriage-wheel bearings is rendered unimportant by the employment of cheap, renewable bushings. The chain has always been one of the most costly portions of the furnace, for though made of hand-welded steel links it is apt to give way by opening at the welds. Chains have lately been made consisting of solid steel drop-forgings for the alternate links, these being connected by steel Ds, one long tongue of which is put through an eye and bent over, so that there is no weld in the entire chain. The consumption of fuel has been considerably reduced with great benefit to the furnace and machinery, and without prejudice to the roast. The cost of erection has also been greatly diminished, while the furnace is stronger and more durable.

The improved Allen-O'Harra calciner is shown in Plate II. The ore is fed automatically from the hopper A on to the upper hearth B, and is gradually moved by the plows toward the further end of the hearth, where it drops through the slot C on to the lower hearth D. It thence traverses the lower hearth until it reaches the discharge E. The chain is driven by the sprocket-wheel F, on the shaft G, and is kept taut by the wheel H in the sliding frame I, which is provided with a weight, J. Six sets of plows, K, are attached at equal intervals to the chain. They are carried on wheels running on the track L. The chain is also supported by simple trucks M, midway between the plow-carriages. It will be noticed that the vanes on the separate halves of the same plow turn furrows in opposite directions; also, that the same plow on the upper floor turns furrows in a direction opposite to its furrows on the lower floor, and that each plow turns furrows in a direction contrary to those made by the plow preceding it. A vane set to turn a furrow toward a guide-rail, or wheel, is fastened to the plow-shaft at some distance to the rail and wheel, so as not to cover the rail, nor to throw ore into the path of the wheel. A



THE ALLEN-O'HARRA CALCINER.

vane set to turn a furrow away from a guide-rail, or wheel, is fastened on the plow-shaft close to the rail and wheel, so as to turn the ore away from them. The arrangement of the vanes on the separate halves of the same plow, by which they turn furrows in an opposite direction, balances the tendency of the plow to be forced off the track on the side opposite to the direction of the furrows, which it would have if the furrows were all turned in the same direction. The hearths are closed at each end by horizontal turnstile doors N, actuated by the moving carriages. The cooling space O for chain and plows is 23 feet in length. The grid P at the driving-end of the furnace is intended for convenience in repairing chain and plows. There are five pairs of fire-boxes, three for the lower hearth and two for the upper, though only one or two pairs are commonly used. The doors R are provided with dampers to admit air to the hearth. The tie-rods that pass through the upper and lower floors are protected by 2-inch pipes, and may thus be easily renewed if burned out.

The costs of calcining in this furnace can be best studied at the Allen-O'Harras, at Butte, Montana, as it is here that they are working on copper ores on the largest scale, and it is here that they were first adapted to the purpose.

It is difficult to offer an estimate of costs that shall seem fair to both the partisans and the detractors of this furnace. The principal cause of this difficulty is the fact that the most important items of cost may be made to vary from 50 per cent. to 150 per cent., according to the care and skill exercised by those in charge of the furnace. These items are the fuel and the repairs. It is very easy to fire in all sets of fireplaces and burn 10 cords of wood per day; but equally good results are now obtained by firing in only one set, and burning but 3.2 cords of wood per day.

Again, a very little carelessness in regulating the heat may damage the chain and running gear to the extent of $100, or more, in a very short time, and augment the repairs to an excessive sum. But careful firemen can be found, and a month's observation of ten of these furnaces convinces me that there is no occasion for damaging irregularities or serious delays.

I think the following table of costs will be found about correct for the Allen-O'Harra furnace, when run with the regularity and skilled supervision that it receives at the works of The Montana Ore Purchasing Company, or the Butte & Boston Mining Company.

The 9 by 90 feet double-hearth calciners at these works average

50 tons each of concentrates per 24 hours. Much of this material is very coarse, some 8 per cent. of it coming from the roughing-jigs, and barely passing a 2-inch ring. The Butte pyrites decrepitates to a certain extent. The average of 150 partial analyses of certain of these concentrates is:

Copper	12.2	per cent.
Iron	31.9	"
Sulphur	41.2	"
Silica	10.6	"
Silver	0.012	" (4.4 oz. per ton.)
	95.912	"

The following table shows the cost of roasting these concentrates down to 8 per cent. sulphur, at the rate of 50 tons per day (100,000 pounds) per furnace. In these works, there is one foreman and one weighman per shift to eight furnaces. One fireman per shift attends two furnaces. One gallon of black oil at 14 cents is used per 24 hours for the machinery of the eight furnaces.

No transportation of ore to or from calciners is included.

COST OF RUNNING ONE ALLEN-O'HARRA CALCINER 24 HOURS, TREATING 50 TONS.

	Total Expense.	Cost per Ton.
Labor—¼ foreman at $4.00		
1 fireman at......... 4.00		
¼ weighman at...... 3.00		
	$5.75	11.5 cents.
Fuel—3.2 cords wood (410 cubic feet) at $4.70 per cord.	15.04	30.1 "
Repairs	2.00	4.0 "
Lights, oil and oiling	0.75	1.5 "
Two horse-power at $0.25 per day, per horse-power	0.50	1.0 "
Interest on cost of furnace, at 6 per cent. per annum	0.92	1.84 "
Totals	$24.96	49.94 "

The power required has been determined by indicator; the fuel, from the wood delivered to eight calciners during a month; the oil, lights, and proportion of labor in oiling, from the actual costs at the works. All these items, as well as the labor employed at the furnace, are easy to arrive at. Also, the first cost of a furnace, which can be checked in various ways.

The only point open to dispute is the cost of repairs. This has been taken from a two years' run. The cost of repairs as given by H. C. Bellinger, superintendent Montana Ore Purchasing Company,* Butte, was only $1 per furnace per 24 hours. This figure was arrived at from new furnaces, only six months in operation, and which had not required many repairs nor new chains. A chain costs about $130, and on heavy sulphides and constant running, should, with due skill and attention, last about a year.

In its construction, the 9 by 90 foot Allen-O'Harra furnace requires 125,000 red brick, 8,000 fire-brick, 36,000 pounds cast iron, 30,000 pounds wrought iron, and 52 perches stone work, more or less.

Including excavation, it costs in Butte about $6,000.

The Pearce turret furnace may be described as a long, narrow hearth, bent around a circle, the circumference of which is a little greater than the length of the hearth, so that the two ends do not quite meet. At this broken part the roasted ore is discharged. The fresh ore is automatically fed from a hopper at the other side of the break, and is gradually stirred and moved forward by rabbles attached to hollow, air-cooled arms, revolving around a stationary, central column. The wall of the hearth forming the inner circle is provided with a continuous slot for the sweeping passage of the two revolving arms, and this slot is closed by an endless steel tape, which revolves bodily with the rabble-arms, being continuously pressed against the slot, so as to mostly exclude the cold air. The entrance of outside air is still further counteracted by the employment of a slight blast under the grate and through the hollow rabble-arms, which balances the tendency of the draught to suck air into the furnace, cools all the exposed iron surfaces, and enables the metallurgist to introduce an accurately gauged quantity of air, for the purposes of combustion and oxidation (900 cubic feet per minute are used at Argo when running on heavy pyritous ores). The inner skewback wall, that is to say, the wall immediately above the flue, is hung from heavy I-beams, whose extremities are supported by the central column, and by the outer walls of the furnace. The bracing of the furnace is exceedingly simple and effective, consisting merely of circular iron bands for the outside, while any distortion is prevented by radial struts, like the spokes of a wheel, between the lintels and the central column. Two or three fireplaces are spaced around the outer circumference of the circle at appropriate points, the entering flame being kept from immediate contact with the ore by short curtain arches.

* *Engineering and Mining Journal*, July, 22, 1893.

The ore is stirred once in 40 seconds, or a total of 540 times during the six hours that it requires to pass from feed to discharge. Of course the time of roasting and number of stirrings can be regulated to suit the requirements of the material under treatment. The greater length of the outer circumference of the hearth as compared with the inner seems to have no ill effect on the result, the roasting being absolutely uniform over the entire width of the furnace, and the length of each individual plow-blade increasing slightly toward the outer circle, so that it can move the ore the slightly greater distance demanded by the increased size of the circle. These plows are simply plates of ¼-inch steel, and last four to six weeks on pyrites containing 40 per cent. sulphur. The rabble-arms that carry the plows are of 5-inch pipe, and last a year. When the plows require renewal, the entire rabble-arm is uncoupled outside of the slot, and withdrawn through a door in the outer wall, a fresh one with plows already in position being at once substituted.

The width of the hearth in the original furnaces is 6 feet, but some are now being built 7 feet wide. The diameter of the enclosed circular space is 19½ feet, and of the furnace over all, 36 feet. The fireplaces project 6 feet further, and the entire furnace can stand in a quadrangle 36 by 42 feet, thus occupying 1,512 square feet.

Plates III., IV., and V. (Figs. 1 to 9), illustrate the Pearce turret furnace. A is the hearth, forming a circle with a wedge-shaped piece removed at B, for the discharge of the roasted ore. This hearth is constructed over the dust-chamber C, through which the gases pass in a direction contrary to that in which they move upon the hearth. D is the first fireplace and E the second one, the gases moving around the hearth to the flue and downtake F, through which they pass to the dust-chamber. The inner hearth-wall has a continuous slot G (Figs. 3, 4, 5) for the passage of the spoke-like rabble-arms H, which have their hub J around the central column I. This column is stationary, and is hollow to admit of the passage of a light blast of air to the wind-box (hub) J. The superior portion of the inner wall and skewback cannot be built up in the usual manner, and is therefore hung from the eight 12-inch I-beams K by means of stirrups *k*, and the cross-beams L. The rabble-arms H are strongly braced by means of the straining rods *h*, and are revolved by the pinion M which meshes into the bull-wheel N. This wheel is centered by the

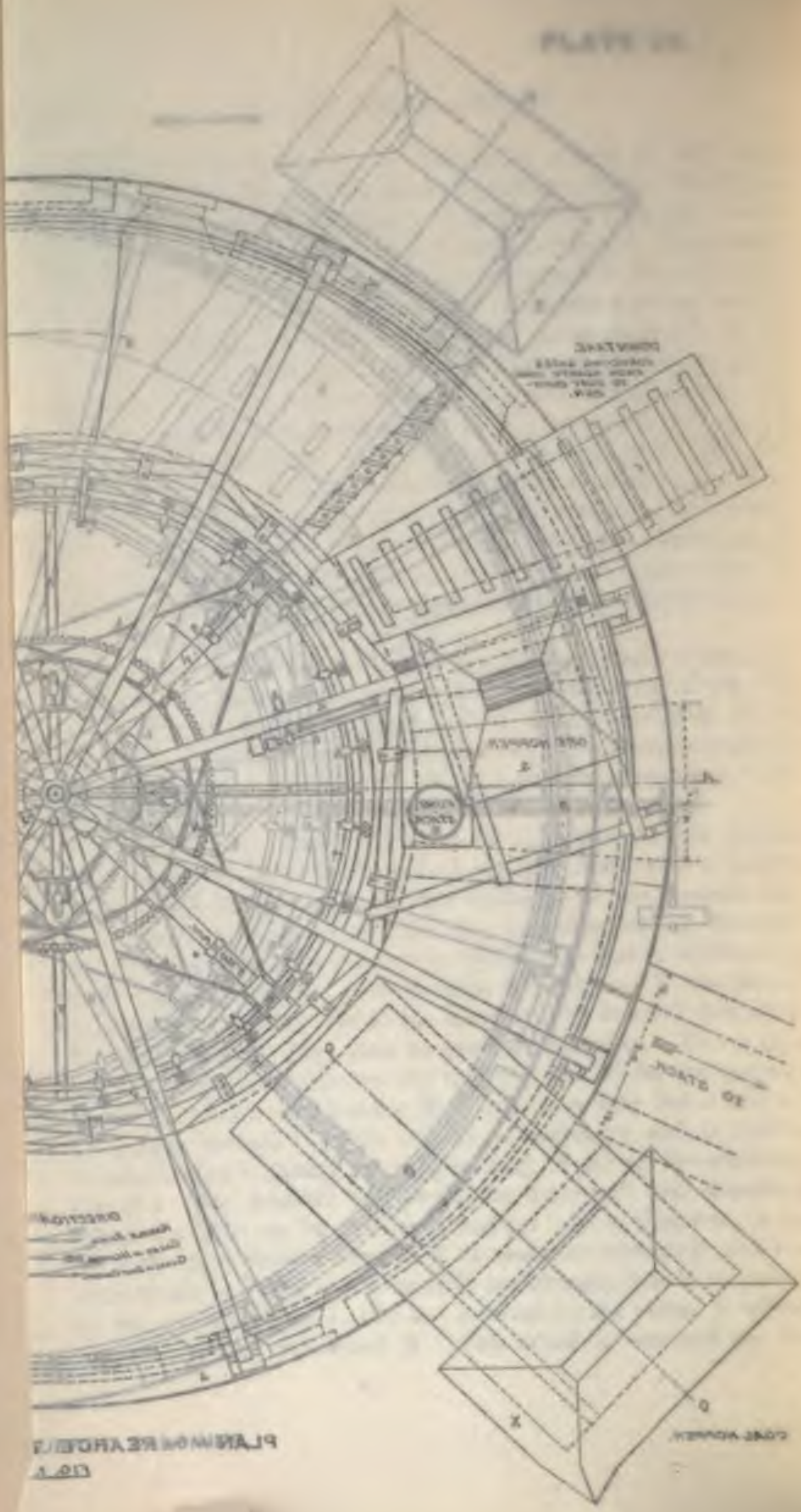
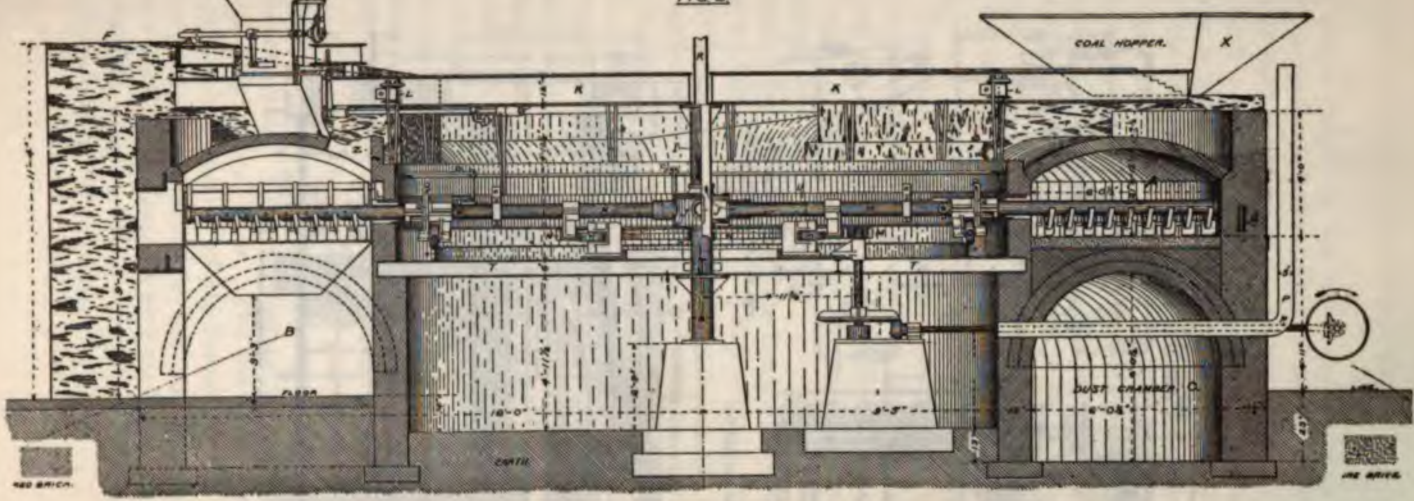


PLATE IV

THE PEARCE TURRET CALCINER.

SECTION ON A.A. FIG. 1.

FIG. 2.



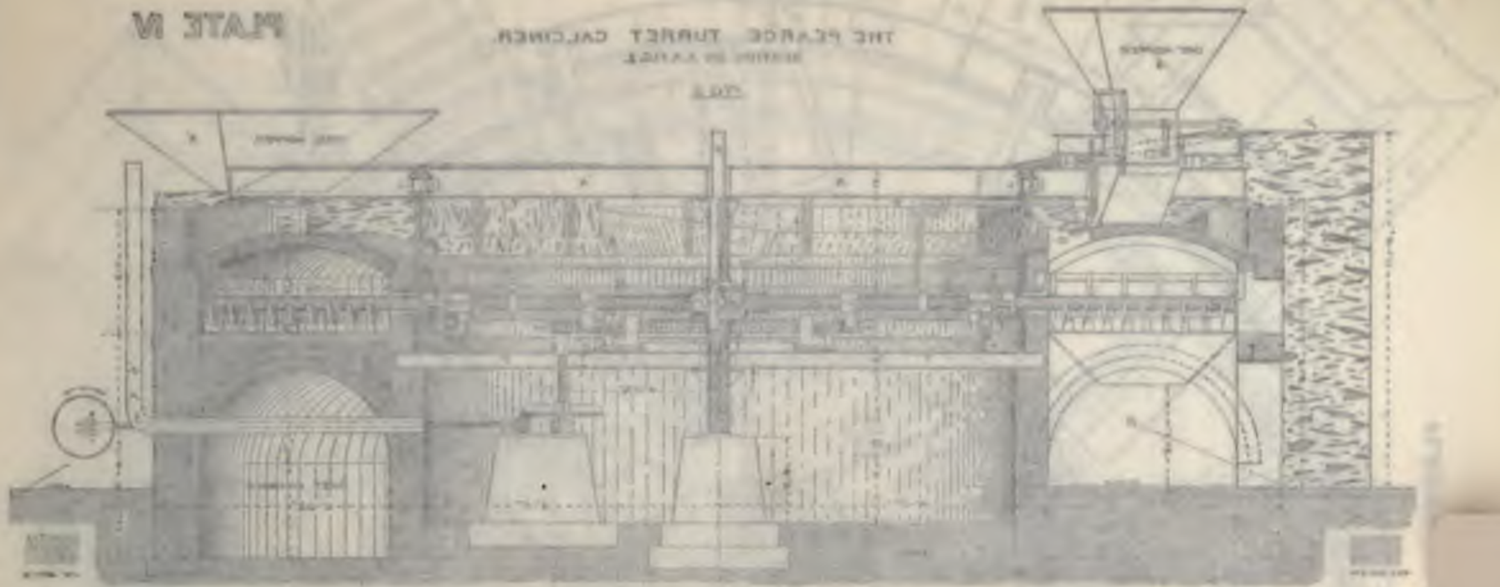


PLATE IV
THE PEARCE TURRET CALCINER.

rollers *n*, and the entire weight of the rabble-arms and driving gear is taken by the conical rollers O running on the circular track *o*; no weight at all comes upon the hub J. A 5-inch pipe P protects the driving-shaft *p* where it traverses the dust-chamber. The rabble-arms have a joint at Q so that they can be adjusted to suit the wearing of the plow-blades. The blast coming through the pipe R, the central column I, and the wind-box J, continues through the rabble-arms H (which consist of 5-inch gas-pipe), and, cooling that portion of the arms which is exposed to the heat of the gases, streams out into the hearth through the openings *h'* and the little pipes *h''* (Figs. 6 and 7), thus cooling the plows and furnishing hot air for the oxidation of the ore. On the first portion of the hearth where the fresh ore is being gradually heated, no air is desired. The blast is, therefore, cut off opposite the ore hopper by means of the butterfly valves *aa* (Figs. 1 and 2), which are closed by the stop *b*, and again opened at *c*. Heated air is also introduced through the exterior wall of the hearth by means of the intramural passages *d*.

The ore is dropped upon the hearth from the hopper S by the automatic feed mechanism shown in Fig. 2 and actuated by the stops *e* on the rabble-arms. It is gradually advanced by the plows in a direction opposite to the gases, until it is discharged at B into a car. The 12-inch I-beams K take their bearings on the central column and on the main outside wall of the furnace. This calciner is strongly banded externally, and is internally braced by the 6-inch struts T that radiate from the central column. The slot G is closed by a 12-inch steel tape U that revolves with the rabble-arms, and is supported and pressed outward against the walls of the slot by means of the bell-cranks and weights *u*, Figs. 8 and 9. The bell-cranks are supported on a circular angle iron V that is bolted to the rabble-arms. The fire-boxes burn slack coal, and are provided with a step grate W, Figs. 3 and 5, and automatic coal hoppers X, Figs. 1 and 2. The fireplace E, nearest the feed, is provided with a curtain arch Y, Fig. 3, as the ore is easily fusible at this stage of the roasting. There are four rabble-arms, but it is found best to use only two of them. The discharge vault is provided with a light stack Z to carry off the fumes.

About two horse-power are required to run the furnace and blast. Apart from repairs and renewals, which are slight, no labor is required at the furnace except to oil the machinery, to fire, and to have a general supervision of its behavior.

Some of the results obtained in ordinary work by this furnace are as follows:

Of iron pyrites containing 43 per cent. sulphur and crushed to pass a two-mesh screen (9 mm. openings), 16 tons per 24 hours are roasted to 6 or 7 per cent. sulphur, using 2¾ tons of Colorado slack coal.

Of matte from the lead smelters, containing 11 per cent. lead, 15 per cent. copper, and 17 per cent. sulphur, crushed through a six-mesh screen (3 mm. openings), 11 tons are roasted in 24 hours to 3.3 per cent. sulphur.

Of concentrated stamp-mill tailings (pyrites), with 45 per cent. sulphur, and 10 per cent. silica, 9 tons were dead-roasted in 24 hours, to show the utility of the furnace for roasting for the extraction of gold by chlorination. No trace of sulphur remained in the roast.

Of Butte concentrates from the Gagnon mine, consisting of variable mixtures of pyrites and blende, but always high in zinc and sulphur, 15 tons per 24 hours are roasted to 6 or 7 per cent. sulphur. The following analysis represents an average sample of these concentrates:

Silica	18.2	per cent.
Iron	20.3	"
Zinc	14.85	"
Copper	11.29	"
Sulphur	31.53	"
Total	96.17	"

The Colorado Smelting and Mining Company, of Butte, has erected double-decked turret-furnaces, the upper hearth of which is supported upon an arch that takes its peripheral bearing upon the main external wall of the furnace, while its inner skewback is supported by the same interior wall that has been already described as hanging from the heavy radial 12-inch I-beams. The inner wall above the slot of the upper hearth being, in its turn, hung from a second set of I-beams, 6 feet higher than the set belonging to the lower hearth.

This double furnace has been only a short time in operation, but excellent results are reported therefrom, especially as regards the consumption of fuel. Each hearth is provided with two fire-places, and Mr. H. Williams, the manager, reports that while the capacity for ore is increased, as might have been expected, from

80 per cent. to 100 per cent., the consumption of fuel is only heightened about 33 per cent. This is an extraordinary and, to me, unaccountable saving in fuel, which I can only explain by assuming that much heat is wasted in the single-hearth furnace; probably because a somewhat high heat is used just before the end of the operation, to partially decompose the sulphates remaining in the roast, and much of it must be lost, owing to the short distance between the third fireplace and the stack.

Indeed, as since the introduction of satisfactory automatic calcining furnaces, fuel has become the main expense in the operation of roasting, it seems a mistake that no more is attempted in the utilization of the heat generated by the oxidation of the pyrites. When we reflect that the heat thus produced is ample to smelt the sulphides themselves, as well as an equal weight of dry ores, and that it is thus utilized in pyritic smelting, we cannot fail to be struck by the seeming extravagance of employing large quantities of expensive, carbonaceous fuel, to burn up Nature's own fuel in the ore. The actual quantity of heat generated by the oxidation of sulphides is exactly the same, whether this oxidation be effected in the pyritic smelting-furnace, or in the calciner. But in the smelting-furnace, it must be oxidized rapidly in order to generate the intense heat necessary for fusion, while in the calciner the oxidation is slow and quiet, being spread over several hours, so as to produce only the moderate temperature suitable for the process. Most of this heat escapes through the stack and in heating the air that is admitted, or finds its way, into calcining-furnaces. The two most obvious means of utilizing this slowly-generated heat, are:

1. By building the hearths in such close juxtaposition that the enormous loss of radiation is lessened, and the waste heat is stored up in the great masses of brick-work forming the furnace. Examples: The improved Spence at the Parrot smelter at Butte, and Steinbeck's multiple-hearth, circular, automatic calciner at Mansfeld, the latter of which runs regularly on argentiferous white metal for the Ziervogel process, absolutely without fuel. The Parrot furnace also runs for days on heavy sulphide ores, at the rate of 30 tons per day or more, with cold fireplaces; and when fuel is used, it is simply to increase the capacity of the furnace.

This type of furnace must not be confounded with furnaces that have their hearths built one above another in what appears to be the same fashion, but where the constructors have taken elaborate measures to carefully isolate and cool each individual hearth. In order to save the possible racking and distortion of the furnace, they sacrifice the main advantage of this method of construction, *i. e.*, the conservation of the heat.

2. By employing the heat of calcination to preheat all air that is to enter either the hearth or the ash-pit. Pearce pursues this plan, to a certain extent, in his turret-furnace, much of the air entering the hearth being preheated by its passage through the rabble-arms, or by passing through canals in the walls of the furnace. Blake carries this still further in his revolving-hearth Cornish calciner at Shullsburg, Wisconsin, preheating the air with the aid of extraneous carbonaceous fuel. He claims valuable results from this system, though it seems a pity to waste coal on preheating the air when such a vast store of heat is available from the operation itself.

Of all the mistaken ideas in the construction of calciners, that of cooling the hearths, except for the purpose of preheating the air used for this purpose, seems to me the most illogical. The occasional disadvantages of distortion can be better borne than the constant waste of fuel. It is like cooling the hearth of a reverberatory smelter by a water-jacket, or by the active circulation of air under a thin hearth, and then wondering why the charges take so long to *bring*, or why they stick so persistently to the bottom. As it is now the fashion to invent automatic calciners, and as the main opportunity in improvement lies in the lessening of the fuel-consumption, it would be most profitable for all aspirants in this direction to spend a week in working at a battery of the pyrites-burners or kilns, as used in the great sulphuric acid works. They would at least learn that the glowing brick-work of the burners is the one kindler, regulator, safety-valve, and balance wheel of the whole operation.

The tendency at present is to drive calcining-furnaces rapidly and burn the sulphur and iron at the highest allowable temperature by means of the heat derived from extraneous fuel, in order to obtain the greatest possible output from a limited calcining capacity.

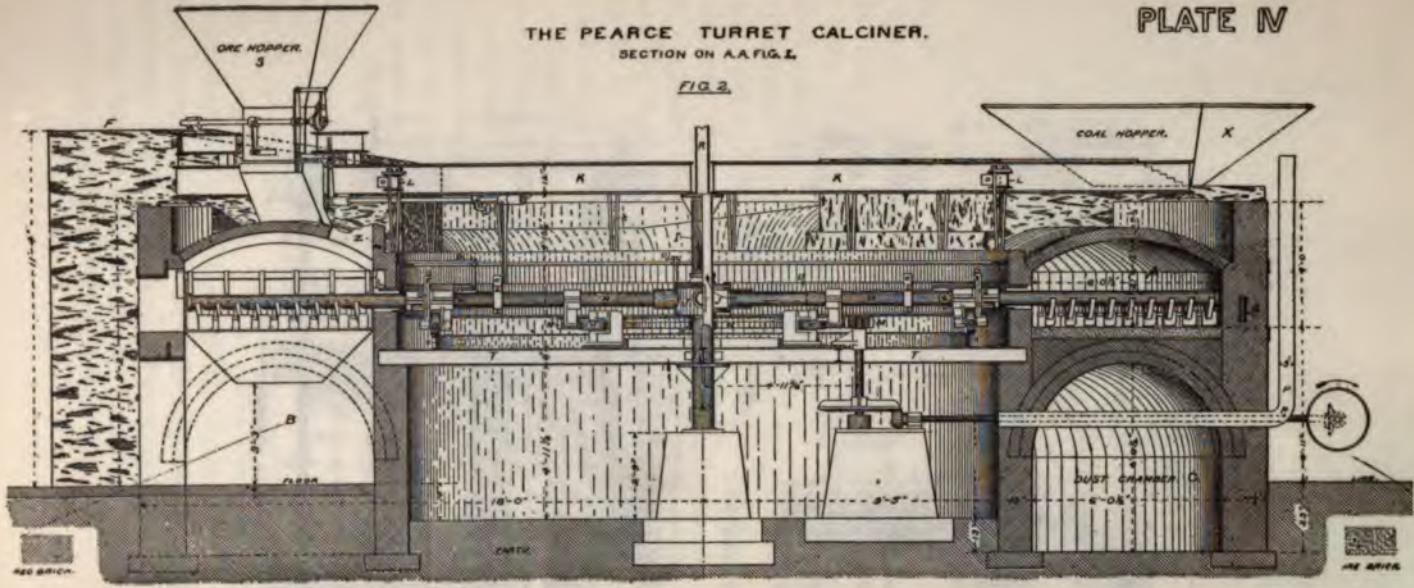
Investment in plant, within reasonable limits, is cheaper than coal at \$3 to \$6 per ton, and it seems probable that slower running, lower heat at the commencement, and through the greater part of the calcining process, and a greater area of hearth per ton of material roasted, will admit of a more thorough utilization of

PLATE IV

THE PEARCE TURRET CALCINER.

SECTION ON A.A FIG. 1.

FIG. 2.



The turret furnace would seem peculiarly adapted to the calcination of auriferous pyrites for extraction by chlorination. Indeed, several are now constructing for that purpose.

In a trial run at Argo, concentrated tailings from the stamp-mills of Gilpin County, Colorado, containing about 79.5 per cent. pyrite, representing 42.1 per cent. sulphur, were roasted down to 0.22 per cent. sulphur at the rate of 9.813 tons per furnace per day. In 8½ days, 83.411 tons were calcined, with the following costs:

TABLE OF COSTS FOR ONE FURNACE, RUNNING 8½ DAYS.

		Total.	Per Ton.
Labor, as before		$23.69	28.4 cents.
Coal—11.64 tons at $2.30	$26.77		
7.981 " 1.75	13.96		
19.621 tons unloading at 8 cents, 1.56		42.29	50.7 "
Repairs—new rabbles, etc.		6.20	7.43 "
Power, steam, and oil		7.42	8.90 "
Interest on furnace at 6 per cent. per annum		7.56	9.07 "
Total cost		$87.16	$1.04.5

Flue-dust.—As may be inferred from the quiet and regular mechanical movements that occur in the turret furnace, its production of flue-dust is very small. In cleaning up the dust-chambers and flues after a run of 2,726.542 tons of ore, 22.65 tons of dust were recovered, being 0.8 per cent.

The cost of a turret furnace at Argo, as built by the inventor, Mr. Pearce, is $5,460.70, inclusive of royalties.

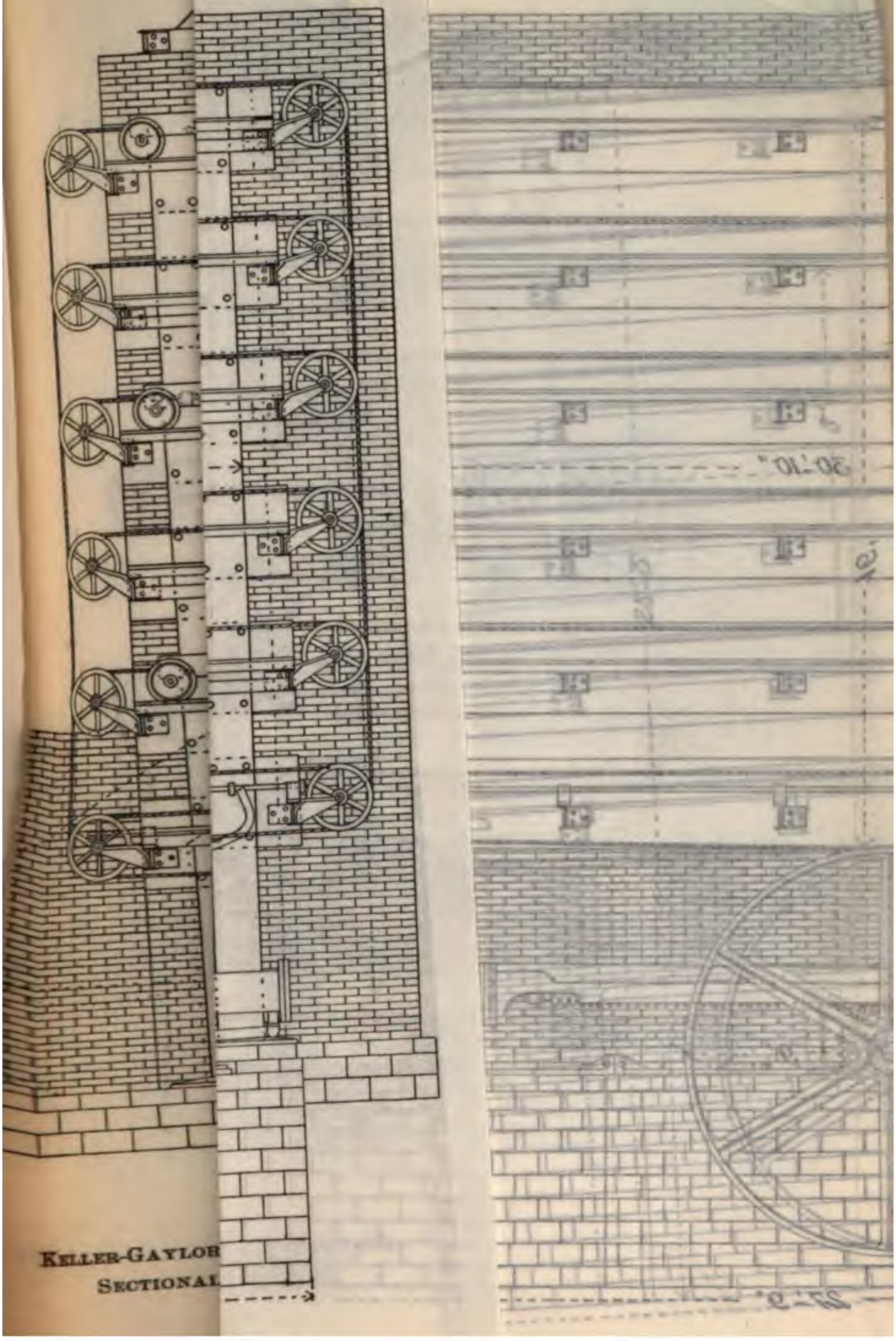
I am indebted to the kindness of Mr. A. S. Dwight, superintendent, for the cost of the two new turret-furnaces erected at The Colorado Smelting Company's Works at Pueblo. The total expense, including royalties, was $12,296, or $6,148 each. In this case there were some extra expenses, owing to necessarily extensive foundations, fire-brick hearths, arches, etc.

The turret-furnace is a model calciner in its running, and in the manner in which its mechanical details have been worked out. It is entirely automatic in its action, one man attending three or more furnaces. It requires but little power to run, and its repair-bill is mainly confined to changing plow-blades once in four to six weeks, and in renewing rabble-arms annually. In only one respect does it seem to me open to criticism, and that is in its consumption of fuel. This is, on good, pyritic ores, some 16 per cent. to 18 per cent. of the weight of the ore; and though it must be remembered that Argo conditions demand a considerably more thorough calcination than is required at Butte, and that it takes more fuel to reduce the sulphur in an ore from 10 per cent. down to 5 per cent. than to lower it from 40 per cent. to 25 per cent., yet there is, nevertheless, too great a loss of heat, and too little use made of the caloric generated by the oxidation of the sulphur and iron in the furnace.

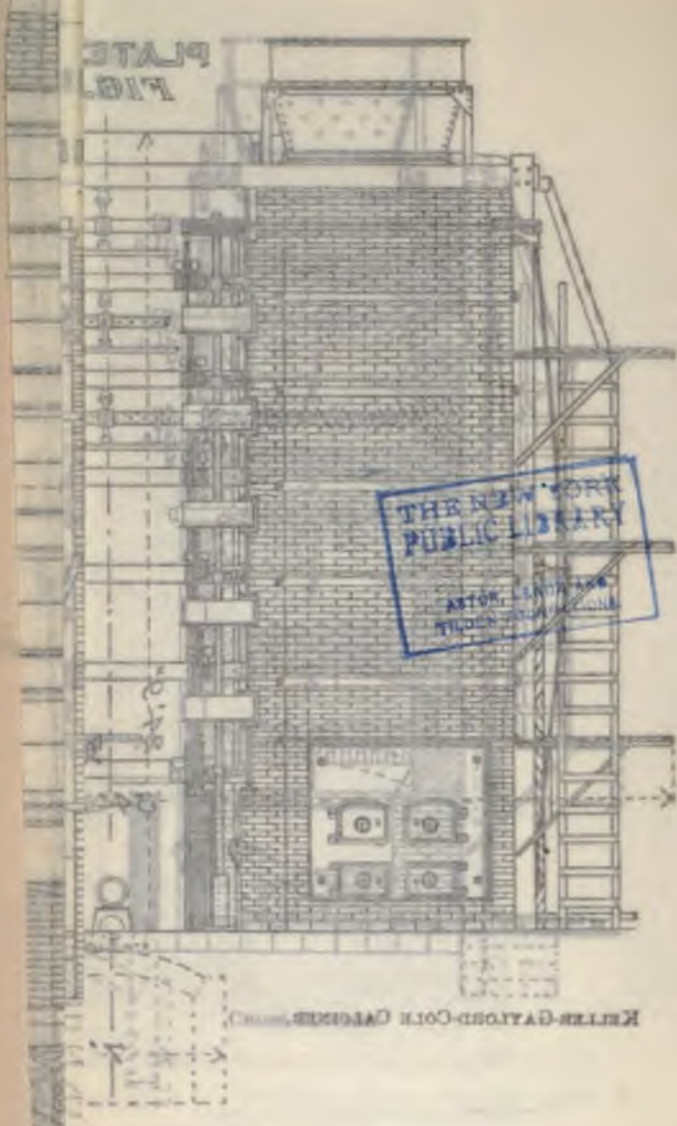
That this is the principal direction in which we must look for a still greater reduction in the cost of calcination is evident, when we note that the fuel, even at the comparatively low price of coal in Colorado, forms about 60 per cent. of the total cost of roasting pyritic ores down to from 4 per cent. to 7 per cent. sulphur.

The improved Spence calcining furnace was designed and erected for the Parrot Silver and Copper Company, by Messrs. Keller, Gaylord, & Cole. The company has lately added two new ones, and now has three of them running at its smelter at Butte, Montana, these having displaced the twelve long reverberatory calciners there in use, as well as the ordinary Spence furnaces which were erected at the Parrot some three years ago. The improved Spence was originally designed as a circular furnace, though the stirring arms returned idle on their track, without ever completing the entire revolution, as in the other circular calciners. But the inventors eventually settled on the present rectangular form, and the furnace is now built as two sets of five hearths and a drying-hearth, the driving mechanism being between these two blocks, and the whole structure constituting a single furnace.

There are, of course, six sets of rabble-arms on each side, one set above the other, projecting through slots into their respective hearths. The rabble-arms are provided with plows both above and below, as in the O'Harra furnace, and these plows are only in contact with the ore when traveling in one direction. When their motion is reversed, a tripping mechanism turns the arm one-fourth of a revolution, so that both its sets of plow-blades lie horizontally above the ore, and in this position the rabbles move back to the other end of the furnace. When they reach this point, the arm is again tripped and revolves 90 degrees. But the revolution of the arm always continues in the same direction, so that the plows that were at first projecting perpendicularly into the air are now brought into use. By this ingenious device the plows are



KELLER-GAYLOR
SECTIONAL



enabled to cool off, and the two sets of plow-blades are so fitted on the rabble that they constantly alternate in the ridges and furrows of the ore on the hearth.

The driving gear consists of a wire rope, the extremities of which are attached to the rabble-frames, while the ropes themselves pass around a large driving-wheel, on whose shaft is keyed a pinion that receives reciprocal motion from a rack actuated by a hydraulic piston.

The slots are closed by traveling steel tapes, as in the turret calciner; but this furnace being longitudinal, and the motion of the rabbles being reciprocally to and fro, the tapes are wound and unwound alternately on horizontal pulleys, placed at each end of the hearth. These are governed by springs so as to keep the tape taut, and its winding is assisted by counter weights.

The hearths are three feet apart vertically, and are covered with siliceous tailings from the concentrator. The enormous mass of brick-work contained in the superincumbent hearths and arches retains much of the heat generated by the oxidation of the sulphides, and consequently diminishes the fuel consumption to a point that would seem impossible to those who have not given attention to this particular subject.

There is a 2½ by 4 foot fireplace, fired with slack coal, to each block of hearths; that is to say, two fireplaces to the double block forming a single furnace. The flame is only allowed to traverse the top hearth, where it is used to ignite the sulphur quickly, the temperature on the lower hearths being ample without extraneous heat to reduce the sulphur to the required standard 7 to 10 per cent. I am informed that by using more time and fuel, there has been no difficulty experienced in reducing the sulphur to any desired limit.

The following results are taken mainly from written statements made to me by Mr. H. A. Keller, superintendent of the Parrot smelter and one of the inventors of the furnace, and therefore cannot carry the same weight as though made by unprejudiced observers. But it is only just to say, that personal observation and careful questioning of the workmen employed about the smelter, especially in regard to repairs, stoppages, and fuel consumption, have failed to detect any exaggeration in the claims made.

The furnace has been mainly run on mixed sizes of concentrates from the Parrot mine, of which the following was the average composition for the first nine months of 1894:

Copper	9.8	per cent.	
Iron	33.8	"	
Silica	13.3	"	
Sulphur	41.2	"	
Silver	0.027	"	(8 oz. per ton).
	98.127	"	

Mr. Keller states that while roasting 45 tons (90,000 pounds) per 24 hours of the above concentrates, the furnace has during the past three months burned three-fourths of a ton of slack coal (at $3.50 per ton). The coal averages about 18 per cent. ash.

The ore is fed to the calciner automatically by heavy fluted rollers; and as the bringing of the raw ore to the furnace, and the removal of the calcined ore depend for their cost upon the general arrangement of the plant, and are, therefore, so variable at different works as to completely invalidate any exact inquiry into the comparative cost of roasting in different types of calciners, I have entirely omitted them in every case, preferring to let each smelter calculate the cost of the above items to suit his individual conditions.

Since the reverberatory calciners have been given up at the Parrot smelter there is no roasting foreman. The three improved Spence calciners are attended by one man per 12-hour shift, who fires (handling ⅜ ton coal for each furnace), and beyond this simply has to oil and oversee the machinery. As his wages are $4 per shift, and the amount of ore handled per shift by the three calciners is 67½ tons, the cost of labor per ton is not quite 6 cents.

Using ⅜ ton coal per shift, at $3.50 per ton, and roasting 22½ tons of ore, the cost for fuel per ton of ore is 5.83 cents. The furnace has been run for several successive days without any fuel at all, the duty being reduced from 45 to 30 tons ore per 24 hours.

It is stated to require two horse-power to run the furnace.

I find from personal inquiry that most of the Butte metallurgists who have carefully followed the development and operation of this furnace seem inclined to admit the correctness of the above statements so far as regards labor and fuel consumption, but are not in a position to express a positive opinion as to the repairs.

I examined the record of the furnace on the Parrot books and found that its stoppages were about 12 hours per month, mainly for renewing rabble-arms and attending to the steel tape that closes the slots.

Mr. Keller's own statements (December 2, 1894), regarding the total repairs on one furnace for the past 12 months, are as follows:

36 sets of plow-blades at $8.94	$321.84
1 full set of 4-inch pipes for arms (12 pipes, each 7 feet long)	29.40
Other repairs, averaging $5 per month	60.00
Total	$411.24

being about $1.13 per day. Mr. Keller calls the repairs $1.25 per day, or 2.78 cents per ton of ore. The rake-end is the only portion of the rabble-arm exposed to heat, and its life, when running 45 tons ore per day, is four months more or less, according to whether it belongs to one of the hotter, or one of the cooler hearths. As they form the main item of repairs, it is interesting to know their cost in detail.

COST OF ONE RAKE-END.

7 feet 4-inch pipe at 35 cents	$2.45
18 cast iron plow-blades, 7 pounds each, at 4 cents	5.04
19 six-inch bolts at 10 cents	1.90
One-half day machine work at $4	2.00
Total	$11.39

It is claimed by the inventors, that there is now no racking of the furnace, nor distortion of slot. There are no fire-brick used in the furnace, except where red brick are so fusible as to be unfit for lining the fire-box.

It will be interesting to assemble the figures already given, and thus determine the cost of roasting at the Parrot smelter, as claimed by Mr. Keller and his associates.

The cost of erecting one of these 45-ton improved Spence furnaces at Butte is about $10,000. The interest on the above sum, at 6 per cent. per annum, would amount to 3.6 cents per ton of ore.

COST OF ROASTING ONE TON (2,000 POUNDS) ORE IN IMPROVED SPENCE CALCINER.

These figures are deduced from H. A. Keller's statements, based on twelve months' running (transportation of ore to and from furnace is not included).

Labor—per ton of ore	6.00	cents.
Fuel " "	5.83	"
Repairs " "	2.78	"
Power and oil, per ton of ore	2.22	"
Interest on cost of furnace per ton of ore	3.06	"
Total	20.43	"

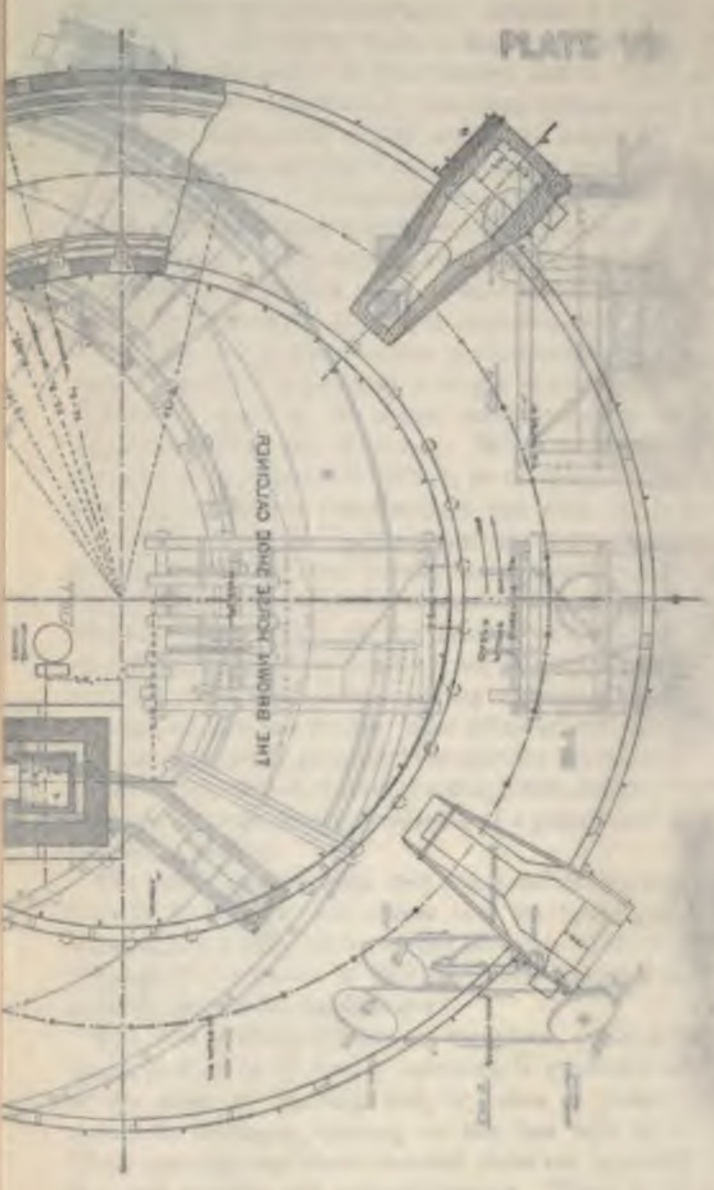
Or about 20½ cents per ton of raw ore.

While these unusually low figures are based primarily on Mr. Keller's own figures at the Parrot smelter, I should not publish them did I not believe them to be, in the main, correct. But a personal examination of the furnace, and a recent visit to the Mansfeld works in Germany, where approximately identical results have been obtained for several successive years in Dr. Steinbeck's modified Parkes calciners (calcining white metal without fuel, for the Ziervogel silver extraction) has enabled me to assimilate these results with less astonishment than many metallurgists will probably experience. The main doubtful point with me is the question of repairs, and on this point I have not had a sufficiently long acquaintance with the furnace to express an intelligent opinion.

The Brown horseshoe furnace is also annular like the turret furnace. But it is bent around a larger circle, the diameter of the unoccupied space in the center being 41 feet 10 inches, and the outer diameter 68 feet 2 inches. With its external fireplaces, it occupies a quadrangle of 73 feet, or an area of 5,329 square feet. The hearth proper is 8 feet wide in the clear and occupies about four-fifths of the circle, the remaining fifth being completely cut out, the free space thus formed being used to cool the rabbles. By means of projecting tiles in roof and floor, a narrow gallery is formed on either side of the hearth. The gallery on the outer circumference contains simply a rail of hard-baked tile, on which runs the outer wheel of the stirring carriage. The inner gallery contains an iron rail for the inner wheel of the same carriage, and also the horizontal, grooved, idler-pulleys which guide the driving cable. This cable is driven by a simplified adaptation of the means employed on cable-roads, consisting of a grip-wheel with tightener and guide-sheaves.

The cable, guide-sheaves and inner rail are cooled by admitting a little air around each sheave into the inner gallery, and it is undoubtedly a valid claim of the inventor, that when the furnace is properly run, none of this iron work becomes hot enough to seriously scorch the naked hand.

The ore is charged from an ingenious, automatic hopper and apron, and, as in all similar calciners, is gradually carried around to the other, or discharge-end, by means of plows, which are attached to carriages, running on the two rails already described. These carriages and their attached plows are intermittently cooled in a very peculiar and original manner. There is always one carriage standing idle on the rails where they cross the open space



etween the adjacent ends of the hearth. It requires about two iinutes for each stirrer to make the circuit of the hearth, so that ie idle one has this same length of time to cool off in. After its nergence from the hearth, the moving (heated) carriage comes i contact with the cooled one that is at rest, pushing it forward short distance, until the carriage in the lead becomes attached › the driving-cable by means of an automatic grip, the heated arriage being detached at the same moment. The Collinsville inc Company of Illinois, and the Glendale Zinc Company of outh St. Louis, Missouri, report, after steadily running the furace for several months, that the action of the grip, cable and heaves is satisfactory, in spite of the high temperature used in asting zincblende.

The Brown horseshoe calciner, as built by Fraser & Chalmers, i illustrated on Plate VIII.

The annular hearth A is broken at B for the ore-discharge, and o afford a cooling space for the plows. These are not shown in he drawing, but are mounted on wheels running upon rails in the iteral galleries C and D, Figs. 1 and 2. The inner rail *c* is of :on; the outer one *d*, of hard-baked tile, except in the broken ortion of the furnace. The plow carriages are moved by an endss cable F, Fig. 5, which runs around the little horizontal rollers ;, and is driven by the ordinary cable-car mechanism, shown in erspective in Fig. 5. The gases flame from the three fire-boxes G, [, and I enters the hearth and passes out through the flue J into the ack *k*. The heated plow which has just completed the circuit : the furnace, comes into the open air at L. It soon comes in ›ntact with the cooled carriage that has been standing in the ›en for some minutes, and pushes it ahead to where it is gripped 7 the cable at M, the heated carriage remaining in the place of ie cooled one. The ore is fed from the hopper N, and is disiarged at H. The rollers E, which are mostly outside of the earth (see Fig. 2.), the cable F, and the rail *c*, are said to be so ›oled by the external air and inward draught as never to reach a mperature of 150 degrees Fahr. (65 degrees Cent.). Air is also lmitted through the roof by means of the holes O. As there are o revolving arms, the hearth is braced with tie-rods in the usual ianner.

It is stated that 1½ horse-power is required to run the machinery. Also that in roasting heavy zincblende ores, about 20,000 pounds of finished product is made per 24 hours, the ore averaging over

30 per cent. sulphur, and being roasted down to 0.85 per cent. to 1 per cent. There are four fire-boxes on the Collinsville furnaces, and about 12,000 pounds of refuse slack from the adjacent coal mines is used per 24 hours.

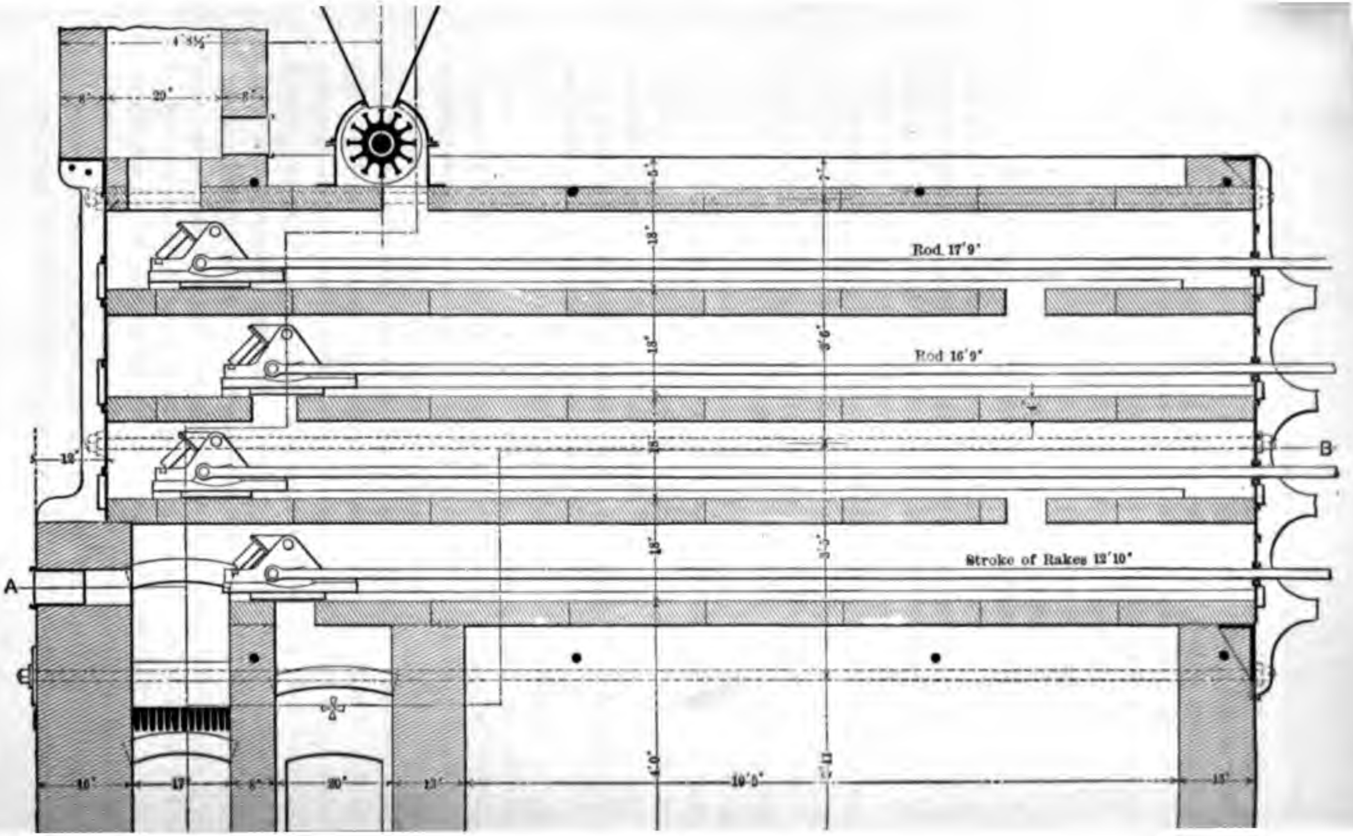
Of course, this is no fair test as to what the furnace would accomplish on ordinary pyritic ores, but there seems no reason to doubt that it will oxidize as rapidly and effectively as any of its rivals under equal conditions.

The Consolidated Kansas City Smelting and Refining Company has just erected one of these furnaces for the sulphate-oxide calcination of its copper-lead matte, for the Hunt & Douglas copper-extraction process. The company inform me that they are roasting mattes for the Hunt & Douglas wet extraction process, and containing 10 per cent. to 20 per cent. lead and 25 per cent. to 35 per cent. copper, at the rate of 36,000 pounds per furnace per 24 hours, with 3½ to 3¾ tons slack coal. The calcined ore contains 87 per cent. to 92 per cent. of its copper soluble in the Hunt & Douglas bath.

Brown urges, as a valuable feature of his furnace, the long road that each particle of ore has to travel. He claims that it is thus peculiarly suited to the roasting of easily fusible ores, as they are advanced so slowly and gradually toward the hotter portion, that the sulphides have ample time to decompose and lose their extreme fusibility before being subjected to a temperature higher than they can bear.

The furnace has stood a severe test in its satisfactory work on zincblende ores for more than a year, and has now entered into competition with the other automatic copper-calciners.

The Spence automatic desulphurizer is a Malétra furnace improved and provided with automatic rakes. It is extensively used and is too well-known to require a detailed description. Fig. 25 gives a longitudinal section in detail. It is used much for roasting fines for sulphuric acid manufacture, but, in its present form, has too small a capacity, and requires too much power per ton of product to compete with the newer automatic calciners. At the Parrot smelter in Butte, Montana, a double Spence furnace has roasted 16,000 pounds of concentrates per 24 hours, reducing the sulphur from 40 per cent. to 8 per cent. This is an unusual duty, and yet is much too small for prevailing conditions. The cost of roasting at the Parrot, in these furnaces, is reported to me to be about $1.25 per ton of ore.


A
B
Rod 17'9"
Rod 16'9"
Stroke of Rakes 12'10"
18"
18"
18"
18"
4'0"
16'0"
3'5"
3'6"
20"
17"
4"

The accompanying drawing shows a Hammond improved Spence furnace used at the great Treadwell mill, Douglas Island, Alaska, where a number of them are employed in roasting the gold-bearing concentrates for treatment by the Plattner chlorination process; six double furnaces roast from 18 to 20 tons of concentrates a day to a "dead roast," with an expenditure of about one-eighth cord of wood per ton of ore. The space required is small and no skilled labor is necessary. Once adjusted, it will continually discharge a finished product. Two men on a shift can attend to six double furnaces easily. One keeps the hoppers full while the other keeps the temperature even. The fronts and backs of the furnaces are so arranged that the supply of ore can be regulated exactly. The dust is even less than in the old reverberatory. A substantial hydraulic cylinder moves the rakes, which are so arranged as to prevent the banking of the material at the ends of the furnace. The iron rails of the Spence furnace, which gave much trouble, are replaced in this by very hard brick tiles. With ordinary care the iron rakes will last six months when salt is used in roasting, and two years when it is not employed, and when burnt out can be replaced by new rakes in ten minutes.

The Mathiessen & Hegeler Zinc Company of La Salle, Illinois, has developed since 1889 a peculiar, but for zinc ores effective type of calcining furnace. In estimating its work, it must be remembered that it is used solely for the sweet-roasting of zinc-blende ores, and that its gases are employed for the manufacture of sulphuric acid, when it is so desired.

It consists of two seven-storied hearths, built side by side in one block, the hearths being 4½ by 46 feet, and possessing a common division wall. The furnace is heated by generator gas, the flame passing back and forth under the three lower hearths, the upper ones receiving no extraneous heat. There is one rake for each double hearth, and this implement rests most of the time on a swinging platform at the end of the hearth. About once an hour, the rake is attached by hand to an iron bar that is pushed through the hearth from the opposite end, and is then dragged back through the ore, the bar being moved by friction pulleys. The outside platform, on which the rake normally rests, can be swung around opposite the opening of the twin hearth, and is then dragged back through the latter in the same manner being thus exposed but a short time to the high temperature of the hearth. The company inform me that a double, seven-story furnace produces 40,000 pounds of thoroughly roasted ore per 24 hours, from 7,000 pounds of zincblende, with a consumption of 9,600 pounds of refuse slack coal.

(*b*) CALCINERS WITH MOVABLE HEARTH.

*Blake** describes a tabular, revolving roaster with automatic feed and delivery, that is said to be an improvement on Brunton's Cornish calcining furnace. It is intended and used for calcining the iron pyrites in the impure zincblende of Shullsburg, Wisconsin, so that it may be easily removed from the blende by mechanical concentration. It could, of course, be adapted for copper ores. It consists of a circular, terraced table, 16 feet in diameter, covered with fire-brick, and made to revolve slowly (10 revolutions per hour) in a horizontal plane. It is supported upon cast-iron balls running in a grooved, circular track 12 feet in diameter, and is covered with a dome-shaped arch. Plows fixed in the roof stir the ore, and gradually urge it downhill toward the circumference of the hearth. Careful arrangements are made for the introduction of pure air, strongly preheated by two Siemens' accumulators. As no assays or analyses are as yet made public, and as the purpose and conditions of the calcination at Shullsburg are totally different from the requirements of the copper metallurgist, it is impossible to institute any comparisons as to results. In calcining the pyrite in a mixture consisting of equal parts of pyrite and zincblende in wheat-sized grains, Mr. Blake states that 20 tons per 24 hours is the regular duty of a 16-foot furnace.

* *Transactions American Institute Mining Engineers*, Vol. XXI., p. 943.

CHAPTER IX.

THE SMELTING OF COPPER.

THE object of smelting ores of copper is to effect a separation of the metal by a mechanical process of concentration, many chemical changes important to the result also occurring before the worthless and valuable portions of the ore can separate according to their specific gravity. The entire mass of rock which contains the copper (often also gold and silver) must be rendered so liquid that the metallic or sulphide portions can freely sink to the bottom, whence they can be drawn off separately, while the worthless molten rock (slag) floats on the surface, and is removed by appropriate means.

In smelting sulphide ores, we cannot profitably produce metallic copper at a single operation; for both the cost of removing *all* the sulphur (calcination), and the tenor of the slag would be too high. The greater portion of the sulphur is removed from the ore by calcination, and the remaining sulphur combines with the copper, and with a certain amount of iron, to form the matte or regulus which is the object of our exertions, and which may be regarded as a highly concentrated ore, free from gangue rock and containing 90 per cent. of the copper, 96 per cent. of the silver, and 99 per cent. of the gold that was present, *by assay*, in the original ore. (Of course these results vary considerably, according to degree of concentration, composition of ores, etc.)

It will be at once apparent, that the higher the degree of concentration, *i.e.*, the more tons of ore we can put into one ton of matte, the lighter will be the future cost of refining this matte, per ton of original ore. For instance: If, in smelting 12 tons of ore, we can throw 11 tons over the dump in the shape of slag, and concentrate the entire value of the 12 tons of ore into one ton of matte, the cost of refining that matte, at $18 per ton, will be divided by twelve, thus being only $1 50 on the ton of original ore. But, if we can only put three tons into one, as often at

Butte, Montana, each ton of ore must be charged with $6 for matte-refining, making a difference in results of $1,000 a day for a smelter of ordinary capacity.

The main factor in determining the limit of concentration is the percentage of copper contained in the original ore. In Butte it is found more profitable (or more rapid) to submit the low-grade ores to a mechanical concentration by water, so that the material that goes to the furnace will already assay 10 per cent. to 20 per cent. copper.

Experience has shown that we cannot make a product at the first fusion going higher than 50 per cent. to 60 per cent. copper, without too great a loss of metal in the slag, and other technical difficulties. Hence, the low ratio of concentration at Butte.

The opposite extreme may be illustrated by the practice at the Argo works in Colorado. This is, commercially speaking, a gold and silver smelter, making use of a very small percentage of copper to collect the precious metals into a rich matte. Regarded metallurgically, however, it is strictly a *copper* smelter; for the minute percentage of silver and gold present have no chemical influence upon the operation. Therefore, we may regard Argo as a copper smelter, treating ores averaging 3 per cent. copper, and, in a single fusion, concentrating 12 or more tons of ore into one ton of 40 per cent. matte. Hence the possibility of the long and intricate series of operations by which the silver, gold, and copper are separated and refined. If it were not for the unusual degree of concentration at the first smelting, this practice would not be a commercial success; and if it were not for the low tenor of the charge in copper, the high concentration would be impossible. Therefore, Argo is not a purchaser of rich copper ores, unless they are very high in the precious metals as well.

I desire to particularly call attention to the fact that, *with a proper slag*, silver and gold may be concentrated in matte to any reasonable extent (by keeping the slag siliceous and tolerably free from zinc, I have gone up to 30 ounces gold and 2,500 ounces silver per ton of matte, without any marked loss), as they do not increase the bulk of the matte or, practically speaking, the percentage of its metallic contents, and thus lessen the percentage of the protecting sulphur to a dangerous degree; but, that the concentration of copper is limited by a figure represented by the percentage of that metal in the highest profitable matte that we dare to make,(that figure usually varying from 35 per cent. to 60 per cent., divided by the percentage of copper in the ore smelted), less one-half to one per cent. for losses in the slag.

For instance, the ratio of concentration for an 8 per cent. copper ore, under conditions where it was most profitable to make a 45 per cent. matte, would be

$$\frac{45}{8-0.5}=6$$

That is to say, six tons of 8 per cent. ore must be smelted to produce one ton of 45 per cent. matte.

It happens, therefore, not infrequently, that there are mines in remote and inaccessible districts, which would be sufficiently rich in gold and silver to yield good profits, were it not that they were too rich in copper. The rate of concentration obtainable by smelting is too low to yield a product of sufficient value to pay the very high transportation charges.

The principal aim of the copper smelter is to get as much of his ore over the dump, in the shape of slag from the first fusion, and to concentrate his copper, gold, and silver into a high-grade matte, as rapidly and perfectly as possible. But there are many complicated chemical changes that must take place in the furnace before this result is obtained, and without a fair knowledge of these important reactions and of certain of the laws of chemical affinity, the smelter cannot have any sound insight into his work, nor any certainty of succeeding when he is confronted with new ores or untried conditions.

Old smelters, who pride themselves on being "practical," should realize that "practical men" usually have infinitely more theories on every subject than scientific men; only they are all wrong.

The most important reactions that occur in the furnace will be briefly enumerated in the description of each method of smelting.

The ordinary products of copper furnaces may be blister copper, black copper, copper bottoms, matte, speiss, slag, and flue-dust.

There are various excellent metallurgical works in which these substances are thoroughly discussed and analyzed. I shall, therefore, merely offer some few practical observations about them that do not find a place in the ordinary text-books.

Blister copper, or more properly, *blistered* copper, is a high-grade crude copper in which nearly all the oxidizable impurities have been removed by slagging and volatilization. Good blister contains from 97 per cent. to 99 per cent. copper and only 0.25 per cent. to 0.75 per cent. sulphur, which, at the high melting point of metallic copper, and in the presence of air, escapes rapidly as sulphurous, and anhydrous sulphuric acid gas. This ebullition of gas continues up to the moment of chilling, and the gas still generated in the molten portion of the pig raises little bubbles and blisters on the surface of the metal, whence its name is derived.

As may be inferred, the production of this material is usually confined to operations conducted with a powerfully oxidizing atmosphere, such as reverberatory furnaces and Bessemer converters. It may, however, under exceptional conditions, be produced in blast-furnaces running on oxidized ores, and, as an experiment, I have produced excellent blister from roasted matte, in the little black copper cupolas at Ely, Vermont, which, for the past 30 years have been run something after the fashion of a pyritic smelter, with a highly oxidizing atmosphere, and producing, ordinarily, black copper of the highest grade. I have seen excellent blister copper produced by Dr. Trippel from oxidized ore in the Longfellow cupolas. This product, when broken, has the true rosy color of pure copper, but not its fine, silken texture.

It is very tough when cold, but its quality of redshortness enables the smelter to separate the pigs of a bed of blister as tapped from the furnace, by breaking the narrow necks that still connect the pigs, the instant that they are sufficiently set to stand the pressure of the bars used in prying them apart.

Black copper is the name given to the more or less impure metallic copper produced in blast-furnaces when running on oxide ores or roasted sulphide material. It is always an alloy of copper with one or more other metals, generally containing several per cent. of iron, often lead, and many other impurities, according to the ores from which it is produced. It usually contains 1 per cent. to 3 per cent. or more sulphur. On cooling, the surface oxidizes, giving it a dull, blackish appearance, nor does its fracture show either the exact color or texture of pure copper.

Copper bottoms is a technical expression, referring to a metallic product of a very indefinite composition, made (usually) in reverberatory furnaces by smelting rich cupriferous substances without sufficient sulphur to quite satisfy the copper present. The affinity of metallic copper for certain substances is much greater than that of copper matte, and the object of employing this smelting for "bottoms" is to cause these substances to combine with a small fraction of metallic copper, by which the main portion of the copper is obtained in a matte freed from them. These alloying substances may be objectionable, as arsenic, antimony, tin, lead, tellurium, etc., or may be highly desirable, as gold or silver.

Matte (regulus) is ordinarily the main valuable product in the first fusion of sulphide ores of copper. Although every metallurgist is extremely familiar with this curious substance, I am at a loss how to define it, as it has but a single essential constituent—sulphur. Without sulphur we cannot have a matte in the sense in which this term is commonly understood. The copper metallurgist would naturally consider copper a rather indispensable constituent of his matte, but the gold and silver sulphide-smelter might make a matte containing no trace of copper, or, possibly, no iron. Nickel, cobalt, lead, or bismuth may take the place of either or both of the metals just mentioned; manganese or zinc may replace them to a marked extent, while those metallurgists accustomed to running heavy-spar ores in cupolas need scarcely be informed that sulphide of barium may become a constituent of the matte to an almost unlimited extent.

But, for the purposes of the copper smelter, matte may be generally regarded as a mixture of cuprous sulphide (Cu_2S) with ferrous monosulphide (FeS) in varying proportions. Thus, in rapid blast-furnace smelting in a cupola with boshes, where the material is calcined ores, or ores containing no bisulphides, and where we can pretty nearly disregard any volatilization or oxidation of the sulphur in the furnace itself, we may consider that each pound of copper present will take up one-fourth of a pound of sulphur, and that the remaining sulphur will take up iron at the rate of about one and three-fourths pounds for each pound of sulphur, all these newly produced sulphides mixing together to form a more or less homogeneous matte.

In less rapid smelting, and where the volume of blast is great, and the shape of the furnace such as to favor oxidation, the amount of sulphur eliminated as sulphurous acid may be very great.* But in steady running, we can usually determine pretty closely our co-efficient of oxidation in each individual case, and should thus be able to determine quite accurately the grade of our matte in advance, were it not for the possible presence of a dis-

*It is this fact that puts into our hands the power of controlling the rate of concentration in blast-furnace smelting. This fact has been long and loudly insisted upon by F. L. Bartlett and Herbert Lang, but, apart from the pyritic smelters, has apparently found few receptive listeners. It will be more fully discussed in other chapters.

turbing element that is so curious and unexpected as to cause many metallurgists to deny the possibility of its existence, until careful and repeated investigations seem to have settled the question. This unlooked for substance is magnetic oxide of iron, which is a frequent, and occasionally important, constituent of mattes. It behaves in a manner that appears at the first glance somewhat paradoxical, for it seems to be formed most persistently and in the greatest quantities in furnaces where there is the strongest reducing action, and where either a contracted hearth and considerable height of ore column, or a large proportion of sulphur in the charge, would seem to forbid the possibility of any oxidizing influence. I have frequently found it in considerable amounts in the matte produced by the rapid smelting of partly oxidized ores in the large type of Rachette furnaces, and have noticed it in still greater proportion in the low-grade matte produced during the quick fusion of siliceous, raw pyrites fines, the charge containing 25 per cent. to 30 per cent. sulphur. It also frequently occurs in lead-furnace mattes in spite of the powerful reducing action resulting from slow smelting, high ore-column, and contraction of the shaft at the tuyeres.

Certain observations of W. L. Austin first assisted me in explaining this phenomenon—to my own satisfaction at least. Austin noticed that in practising pyritic smelting with small tuyeres and a high blast pressure, the partially, or entirely, molten sulphides, as they dropped in front of the tuyeres and received the full force of the blast, were often in part changed to magnetic oxide, a cauliflower-like excrescence of this oxide forming almost instantaneously on the surface of a partially fused mass, and this in spite of the proximity of a great preponderance of vaporous sulphur and sulphurous acid. This may well be the origin of much of the magnetic oxide in the instances that have come under my own notice. Being a feeble base and of high specific gravity, it does not combine with the silica, but settles to the bottom, mixing with the matte and becoming a part of the latter. This formation of magnetic oxide of iron is generally an unfortunate circumstance, doing harm in at least five different ways:

1. It robs the slag of the iron that is needed for flux.

2. It lessens the dissolving power of the matte for silver, and perhaps for gold.

3. It increases the quantity of matte to be treated later.

4. It makes the matte exceedingly tough and tenacious, and expensive to break or pulverize.

5. It makes the charge less fusible.

If our theory of this formation of magnetic oxide of iron be correct, it is very easy to suggest the remedy. It is not too rapid nor too slow fusion, nor too much nor too little reduction that causes the formation of magnetic oxide. It is simply too high wind pressure; and that this circumstance seems to stand in close relation to its production is shown by the fact that, in the cases that I have just referred tô, the production of this unwelcome oxide diminished greatly, or ceased completely, with the lessening of the blast pressure. But this modification of practice means something more than simply reducing the blast pressure; for if this alone were done, the capacity of the furnace would probably fall off to an extent that could not be tolerated. The powerful blast that was used conduced to rapid smelting and great capacity, and also presupposed tolerably small tuyeres and a furnace shaft of considerable diameter, or width; probably 40 to 48 inches. The weakened blast now proposed cannot successfully penetrate the ore column in a shaft over 34 inches in width, and this may, in some cases, much better be reduced to 30 inches, and the proper capacity retained by enlarging the furnace in the only dimension possible, that of its length.

This gives us a long, narrow rectangle, and, as we are obliged to decrease our wind pressure, we must enlarge our tuyeres, in order to obtain a sufficient volume of air to burn the considerable quantities of fuel that fill this space. The low pressure and large volume of blast required suggest at once the employment of a large fan blower in place of a positive, or semi-positive, blast machine, and, if it were not for the annoyance caused by large belts driving small pulleys at a high speed, I should feel much inclined to return to the stand taken some years ago by Mr. H. M. Howe in regard to fan-blowers.

In a work like the present one, devoted almost exclusively to the practical side of metallurgy, it is impossible to even enumerate all the interesting questions still presented by matte for study and experiment.

Is it a chemical combination, a mixture, or a partial alloy?

What are the affinities of the various sulphides that it may contain, at smelting temperatures, and how do they vary among themselves as the temperature rises and sinks?

What affinity or power of alliage is there between the metallic sulphides and those of barium and calcium?

Why does the same matte separate more quickly and thoroughly from an acid slag than from an equally light, and much thinner, basic slag (containing principally alkaline and earthy bases)?

Why does the capacity of matte to collect the silver of an ordinary charge increase to a certain point as its copper contents increase, and then retrograde as the matte becomes still richer in copper, while its affinity for gold continues increasing, metallic copper having the greatest affinity of all?

Why does a cone of matte, allowed to cool naturally, crack parallel with its surface when containing over 50 per cent. copper, and at right angles to this direction when below 50 per cent. copper?*

These are but a few of the unexplained phenomena regarding matte that are constantly forcing themselves on the copper smelter's attention.

Speiss, as ordinarily understood, is a basic arsenide, or antimonide of iron, often with nickel, cobalt, lead, bismuth, copper, etc., having a metallic luster, high specific gravity, and a strong tendency toward crystallization. It takes up gold with avidity, but has a less affinity for silver than copper matte has.

It has always seemed to me that here is a field that has not been sufficiently exploited. Especially since bessemerizing and pyritic smelting are becoming so important, it is worth while to consider to what degree, and with what advantages, speiss may be used to replace sulphides under favorable conditions. We have several instances where it has been used to collect silver, gold, or copper. A late notable example in the Transvaal, South Africa, of which, I regret to say, I have no personal knowledge, is described by Mr. W. Bettel in the *Chemical News* of June 26, 1891. He describes the production of an argentiferous, antimonial copper speiss of the following composition, from smelting oxidized, ferruginous ores, containing much antimonate of iron, and 4 per cent. of copper in the shape of carbonates, and 36 ounces silver per ton (0.123 per cent.).

* This fact was first pointed out to me by H. C. Bellinger at the Montana Ore Purchasing Company's smelter at Butte, Montana.

Copper	52.50
Antimony	38.00
Arsenic	2.00
Sulphur	2.06
Iron	3.60
Silver	1.59
Lead	0.25
	100.00

The ore is smelted in reverberatory furnaces, and some 91 per cent. of the silver and copper is collected in the speiss. The concentration averages 16.4 tons into one.

Slags.—The copper metallurgist approaches this subject from a totally different standpoint from that of the lead-silver smelter. It has been shown by many able writers that to obtain slags low in lead and silver, it is advisable in lead smelting to form the slag so that there may be some definite and constant ratio between the iron, lime, and silica that form its principal constituents. After numerous experiments under varying conditions, I am unable to detect any such law that can be applied to copper matte slags. From a considerable number of determinations, I select the following, the chemical work of these experiments having been mostly done by Messrs. D. Murphy, A. R. Vincent, and T. G. Rockwell. In all the cases the sampling was conducted with care, a small ladleful of slag being caught under the slag-spout just as each pot was pulled away, while equal pains were taken to obtain a true sample of the matte. Each separate type of slag was run for six hours, and no samples were taken of molten material from the fresh charge until it had been in the furnace double the time necessary to reach the tuyeres. Then the furnace and forehearth were tapped completely dry, and sampling was begun after the fresh flow of products had become well established. The furnace part of such experiments is very easily and cheaply done, as it is only necessary to add or subtract a certain calculated portion of siliceous, or basic ore, at each charge.

TABLE SHOWING CONTENTS OF VALUABLE METALS IN CUPOLA SLAGS WITH VARYING PROPORTIONS OF SILICA.

		Slag.					Matte.		
		SiO_2.	FeO.	BaO	Cu.	Ag. Oz.	Cu.	Ag. Oz.	Au. Oz.
Roasted pyrite, siliceous dry ores, and heavy spar ores.............	Normal charge..	29.5	57	7.2	0.38	3.5	38	625.1	3.57
	Increased silica..	33.1	52.9	6.1	0.31	3.15	39.9	601	3.78
	Increased silica..	36.7	45.2	8.6	0.44	2.9	43.3	586.7	4.31
	Diminished silica	25.2	57.6	12.2	0.78	4.4	35.6	666	3.13
Siliceous roasted ore and roasted concentrates....................	Normal charge..	27.6	64.4	CaO	0.61	0.71	54.2	25.1	....
	Increased silica .	30.1	62.1	....	0.82	0.91	56.6	28.4	....
	Increased silica..	32.3	61.1	....	0.77	0.8	56.9	32.2	....
	Increased silica..	35.6	57.7	....	0.61	0.56	58.7	33.4	....
	Increased silica..	38.3	55.5	....	0.6	0.48	59.4	31.4	....
	Diminished silica	24.2	undet	....	0.98	1.86	50.5	22.1	....
Rich, siliceous, argentiferous and zinciferous dry ores, roasted argentiferous pyrite, and a limestone containing copper glance..	Normal charge..	32.5	52.1	11.7	0.31	1.9	32.7	32.2	....
	Increased silica..	35.4	50.7	9.4	0.3	2.2	28.6	384.7	....
	Increased silica..	39.3	46.9	8.1	0.31	1.7	25.5	414.4	....

In the above table everything is given in percentages, excepting the gold and silver. These are given in ounces per ton of 2,000 pounds. To reduce this to percentage, multiply the ounces per ton by 0.003436.

The above results were selected for publication as being among the most uniform and complete of a considerable series of similar tests, but I can detect nothing in them, or in any of the figures obtained, to show that the freedom of a copper slag from valuable metals stands in any especial relation to the stochiometrical proportion or arrangement of its constituents.

We feel, therefore, comparatively untrammeled as to the composition of our slags, providing always that they are sufficiently fusible and that their specific gravity is not so great as to hinder the settling out of them of the matte particles. In planning a new slag, we are, within reasonable limits, guided by commercial rather than by chemical influences, and are tolerably independent of the limestone quarry. That this is peculiarly the case in Pyritic Smelting will be seen when that subject is reached. Nearly every copper metallurgist begins his furnace work by trying to make as basic and ferruginous a slag as circumstances will permit, and finishes by making his slags as siliceous as possible. While skill and good settling facilities may succeed in making a tolerably clean slag from a basic charge, it is very much easier and surer, and need not necessarily take a pound more of coke, to make a quite siliceous slag. This is especially the case where copper is scarce, and the minute proportions of tellurium, bismuth, and other comparatively unstudied substances that so increase the power of the matte to collect the precious metals, are wanting. So far as my own experience goes, I consider an acid slag in such a case, an absolute *sine qua non*.

In the smelting of sulphide ores, unless some unusual conditions prevail, the copper, silver, and gold contained in the slag are present in the shape of shots or prills of matte. Most of these particles are extremely minute and can be best seen by reflected light, and with the aid of a good magnifier. There is no excuse for this condition of things, if it at all exceeds the customary limits. Either the slag must be unsuitable in consistency or gravity for the separation of the matte globules, or, what is very much more common, the settling facilities are inadequate. Especial attention will be paid to this important subject when we come to consider the construction of furnaces.

"What is the best slag to make under my conditions?" is rather a commercial than a metallurgical question. Pretty much anything, within wide limits, can be smelted, and if it is more profitable to produce a slag containing 2 per cent. copper and 10 ounces silver than it is to flux the charge so as to save those metals, the former is the proper slag to make. These abnormal conditions become more and more rare as the Western country is opened up by railroads, but they still exist; and in portions of Mexico may continue to prevail for many years.

The usual object of smelting a copper ore is simply to divide it into two portions: a small quantity of matte for further treatment, and a large amount of slag to go over the dump. Now it is entirely immaterial how this object is accomplished, or whether the ore has been thoroughly fused or only half melted, providing that the work has been done in the cheapest, quickest, and most effective way possible under the circumstances. For instance, the Swansea smelters long ago found out that it did not pay them to flux all the silica when running on a highly quartzose charge. A reverberatory slag may contain close on to 50 per cent. of unmelted fragments of pure quartz, and yet be clean and satisfactory; the main requirement being that there shall be a sufficient proportion of molten slag to float the unfused particles, and enable the worthless portion of the charge to be dragged out of the furnace without carrying with it the valuable part. This species of liquation may at times be used to great advantage.

Flue-dust.—The main practical interest attached to this product is connected with the methods for its collection and treatment, which are considered elsewhere.

For practical purposes we may distinguish three totally separate and distinct methods of smelting:

(*a*) Blast-furnace smelting with carbonaceous fuel. Suited to every class of copper ore, whether metallic, oxides, or sulphides. Atmosphere in furnace, reducing.

(*b*) Reverberatory smelting. Mainly for sulphides. In a subordinate degree, for metallic, and oxide ores. Atmosphere in furnace, neutral.

(*c*) Pyritic smelting.* For sulphide ores, though oxide, or metallic ores may always be added when there is an excess of sulphide. Atmosphere of furnace, oxidizing.

* By the term "Pyritic Smelting," I intend to designate that distinct and characteristic process by which sulphide ores are smelted, in the main, without the use of carbonaceous fuel, the necessary heat for the operation of smelting being obtained from the combustion of the sulphur and iron contained in the ore itself. See chapters xiv and xv.

Vide Hofman - p 274 et seq:

gulo Silicate - - $2\overset{\text{II}}{R}O+SiO_2$ or $2\overset{\text{III}}{R}_2O_3+3SiO_2$

" - $2\overset{\text{III}}{R}O_3+3SiO_2$

qui " - - $2(\overset{\text{II}}{R}_2O_3+3SiO_2)+2(\overset{\text{III}}{R}O_3+3SiO_2)=4\overset{\text{II}}{R}O+3SiO_2$

or $4\overset{\text{III}}{R}_2O_3+9Si($

eO = MnO $FeO \times \frac{7}{9} = Fe$ $Fe \times \frac{9}{7} = FeO$

:aO = MgO × 1.4 CaO = BaO × .4 CaO = ZnO × .7

CHAPTER X.

THE CHEMISTRY OF THE BLAST-FURNACE.

THE one distinctive feature of the blast-furnace is the absence of a separate fireplace, the ore and fuel being in direct contact in their passage through the furnace. It is also, in a more general way, characteristic of it, that its operation is continuous, and that it is provided with a forced blast.

This rapid combustion of carbonaceous fuel produces a strongly reducing atmosphere, and brings about a series of reactions that, although possessing much similarity to those that occur in the neutral atmosphere of the reverberatory furnace and in the oxidizing atmosphere of the pyritic smelter, yet differ in some few details that are of the utmost commercial importance to the smelter of copper ores.

A thorough familiarity with the chemical reactions that occur in the operations of calcining and smelting is, next to natural common sense, the most important attribute of the metallurgist.*

In describing the most striking reactions that take place in the blast furnace, we may assume the charge to consist of calcined pyritous ores of copper, containing a little gold and silver, and sufficient iron, lime, and silica to make a proper slag. The fuel shall be ordinary coke, though, in the main, the same reactions will occur with charcoal.

* In attempting to make the most important of these reactions clear to those who have not had a scientific training, I must necessarily speak in general terms, avoiding such considerations as the influence exerted by the ash of the fuel, the frequent occurrence of oxides of iron in the matte, the presence of sulphides of calcium and iron in the slag, the imperfect working of Fournet's law regarding the order of affinity for sulphur possessed by the various metals, etc. These matters will be treated of in their appropriate place.

We may divide the constituents of the ore into four classes:

Bases.	Protecting Agents.	Reducing Agents.	Acids.
Iron.	Sulphur.	Coke.	Silica.
Lime.	(As.)	(Sulphur.)	(Al_2O_3.)
Copper.	(Sb.)		
(MnO.)	(Te.)		
(ZnO.)			
(MgO.)			
(BaO.)			
(KaO.)			
(NaO.)			
(Al_2O_3.)			

The tendency of the reactions that occur is for the bases either to be reduced to a metallic condition and to be separated out as metals, or to become oxidized and to combine with the silica to form a slag.

Neither of these conditions would satisfy the smelter; for in the one case he would obtain an alloy of metallic iron and copper (lime being a very powerful base, in the main runs no risk of being reduced, but combines with silica without any particular care being required), while the other alternative would be that he would oxidize and slag a considerable portion of the copper present. It is essential, therefore, to steer a middle course between constructing and running a furnace in such a fashion that its reducing action shall be powerful enough to reduce the iron, as well as the desired copper, to a metallic condition; and the opposite extreme (theoretical), where much of the copper would be oxidized and slagged. To maintain this delicate equilibrium would be somewhat difficult (though tolerably attained in the smelting of purely oxidized ores), were it not for the presence of a powerful regulating and protecting agent, in the shape of sulphur.

This element has a very strong affinity for copper, but, under the circumstances that we are considering, can only combine with it when the copper is in a metallic condition. Therefore, the sulphur that is still present in considerable quantity in the imperfectly calcined ore, aided by the powerful reducing gases resulting from the burning of the coke, reduces to its metallic condition such copper as is present in any oxidized form. In so doing, a portion of the sulphur itself is burned by the oxygen that it takes from the copper, and escapes as a gas (SO_2). The carbonic oxide is the main reducing agent, but it is very desirable to be familiar

Atom
Fe
FeO
Fe_2O_3
Cu
MnO
ZnO
MgO
BaO
KaO
NaO
Al_2O_3
Al
Sb
As
Bi
Cd
Ca
Co
Au
Pb
Ni
Hg
Pt
Si
Ag
S
Sn
Ti
Zn
Te
Se

also with the reducing effect of sulphur upon metallic oxides, from which springs the important metallurgical principle, that sulphur and oxide of copper smelted together yield metallic copper and sulphurous acid gas.

The remainder of the sulphur combines with the copper that has just been reduced to a metal, taking it up in about the proportion of four pounds of copper to one pound of sulphur. If only enough sulphur were present to exactly satisfy the copper, the resulting matte would be a pure subsulphide of copper (Cu_2S), containing 80 per cent. and 20 per cent. sulphur. The production of so high grade a matte would not only make the slag too rich in copper, but would render the management of the furnace more difficult; for the constant presence of a considerable quantity of matte of a moderate tenor in copper keeps the furnace and fore-hearth open and hot, and facilitates rapid driving.

In ordinary work, there is no danger of any such contingency, as the calcination of sulphide ores is almost invariably under, rather than overdone.

Hence, there is nearly always more sulphur present than is needed to saturate the copper in the proportion of one pound of sulphur to four pounds of copper. This excess of sulphur proceeds to attack the metal for which it has the next greatest affinity after copper. This metal is iron, which combines with sulphur in the proportion of one and three-fourths pounds of iron to one pound of sulphur. The resulting monosulphide of iron has the property of mixing with subsulphide of copper in all proportions; and the resulting mixed sulphides, being much heavier than the slag, separate therefrom and sink to the bottom.

It must be self-evident that the grade of the matte will depend upon the amount of sulphur present; for after a certain portion of the latter has been burned in reducing the oxide of copper to metal, and a still further portion has combined with the copper to form a subsulphide, every pound of sulphur that is left, and that is not burned in some way, will take up one and three-fourths pounds of iron; thus diluting the matte to the extent of two and three-quarters pounds of worthless sulphide of iron for each pound of superfluous sulphur present.

We have already seen that it was necessary to dilute our matte to a certain limited extent with sulphide of iron, that it might not be too rich. But any sulphide of iron in excess of the amount required to lower the matte to the grade that is found most advantageous for our own local conditions, will usually mean a heavy loss in two directions.

1. It makes an excessive quantity of low-grade matte, thus entailing heavy expenses for its future treatment.

2. It robs the slag of the iron that is usually needed as a flux for the silica present, and carries it into the matte, where it is not wanted.

All this trouble arises from an excess of sulphur in the blast-furnace charge, which, of course, means that there has been an insufficient calcination. When smelting with carbonaceous fuel, the secret of the economical treatment of sulphide ores lies in the calcining furnace.*

Thus far it has been convenient to regard matte merely as a mixture of subsulphide of copper with monosulphide of iron. But, in practice, we rarely find its composition so simple. Indeed, I cannot give a definition of matte that is at all satisfactory to myself. The subsulphide of copper seems to be the most regular and constant basis to start from, but this may be replaced, in whole or in part, by monosulphide of iron, or by the sulphides of nickel, cobalt, lead, manganese, or bismuth, while silver, gold, tin, platinum, iridium, molybdenum, and cadmium are collected in this substance, when they occur in the ores.

Nor is this variability confined to the electro-positive elements. Sulphur is frequently accompanied, or partly replaced, by arsenic, antimony, tellurium, or selenium, all of which combine with the copper, iron, etc., forming frequently a matte of such complexity that it is impossible to construct any formula for it, even after the most careful analysis.

Metallic copper, iron, and lead, and magnetic oxide of iron are also found in mattes, but I cannot regard them as proper constituents of the same. They seem to me either as substances produced by certain reactions inside the furnace, and merely mechanically mixed with the matte, or else to have been in combination with the sulphur, or other metalloids, during the time of fusion, and to have separated out on cooling.

The sulphides of calcium and barium are also, according to my

* Certain conditions may render it more economical to smelt the ores raw, and throw the bulk of the work onto the subsequent converter process. This may be regarded as simply deferring the calcination to a later stage of the process. It will be remembered that we are not considering "Pyritic Smelting" at this time.

observation, merely admixtures, as they will, under proper conditions, separate and float on the surface of the heavier sulphides. They do harm in three ways:

1. By lessening the power of the matte to dissolve the precious metals.

2. By lessening the specific gravity of the matte, so that it will not separate so perfectly from the slag.

3. By carrying into the matte, where they are not wanted, bases that are usually much needed in the slag.

Assuming the slag to be well melted and sufficiently fluid, the action of specific gravity is the sole agent which causes the separation of the matte therefrom.

Hence, it is obvious that, other things being equal, a heavy matte and a light slag would cause the least losses of metal. But as we usually have to put up with ferrous oxide as our principal base, we necessarily produce a slag of too high a specific gravity for the most favorable separation of the matte, and consequently are obliged to adopt extensive settling apparatus, and also to put up with a more or less serious loss of values. Yet, as will be explained more fully in its proper place, much of this loss may be avoided, even with heavy slags and a light matte, providing that the slag is kept very hot and liquid during the settling operation, that the particles of matte are, as far as possible, brought in contact with a larger body of molten matte already settled, and that sufficient time is given for the slow subsidence of such globules of matte as have escaped the contact already referred to.

EXAMPLE OF CALCULATING A BLAST-FURNACE CHARGE.

As I have received a considerable number of requests to give a detailed example of a convenient method of calculating a smelting mixture, I introduce it at this point to illustrate the principles that we have been considering.

It is a late actual case, with figures evened and simplified a little, and although it refers to a raw smelting of the sulphide ores and a subsequent oxidizing fusion of the matte, to fit it for the converter process, it is peculiarly suited to illustrate the reactions mentioned in this chapter, the ore being unusually simple and pure.

The ore that we will take as a practical example shall consist of a mixture of copper, and iron pyrites, in a slaty and abundant gangue. A considerable portion of the chalcopyrite is sufficiently massive to be cheaply picked out by hand as a siliceous first-class ore. The remainder, and by far the greater proportion of the ore, is to be subjected to a mechanical concentration. Although it is not a good ore for the purpose, and test runs have shown that it will undergo a heavy loss, its abundance and extreme cheapness of mining, and the lack of suitable basic ores for flux, render it cheaper to waste a certain proportion of the metal than to save it. The occurrence of a moderate amount of gold and silver in the ore also bars the employment of a wet method. It would be superfluous to go into more detailed explanations of the reasons for adopting the method of treatment to be discussed, as this is in no wise the object of the example.

To make matters plain from the outset, I will begin with the ore as it is delivered at the mouth of the shaft.

MECHANICAL CONCENTRATION OF THE ORE.

We will assume that the mine delivers daily to the concentrator, 600 tons (1,200,000 pounds) crude ore, averaging 4.6 per cent. copper.

	Pounds Cu.
600 tons ore contain..	55,200

Products of hand picking:

	Pounds Cu.	
90 tons waste rock 1.5 per cent.... =	2,700	(4.9 per cent. of original Cu.)
80 tons siliceous selected ore, 10 per cent.................. .... =	16,000	
430 tons ore for concentrator, 4.244 per cent...................... =	36,500	
600 tons. Total..................	55,200	

Products of concentrating mill:

We start with 430 tons ore, 4.2442 per cent. Cu..	= 36,500

Loss in concentration = 40 per cent. on this ore.
We produce

320.5 tons tailings, 2.2777 per cent. Cu =	14,600	(26.5 per cent. of original Cu.)
109.5 tons concentrates, 10 per cent. =	21,900	
430 tons. Total.....	36,500	

Total loss of copper thus far = 31.5 per cent. on original amount.
The products to go to the smelter are, therefore:

109.5 tons pyritous concentrates, 10 per cent., containing	21,900	pounds copper.
80 tons siliceous selected ore, 10 per cent., containing	16,000	" "
189.5 tons in total, containing.......................	37,900	" "

SMELTING.

The smelting treatment is to consist of three operations:

1. Smelting the raw (and tolerably granular) concentrates, with the major portion of the raw, siliceous selected ore, and with the converter and matte-concentration slags, in blast-furnaces, with coke, for a matte with about 30 per cent. copper, and a slag with about 33 per cent. silica.

2. An oxidizing (pyritic) smelting of the raw matte produced in No 1. operation, with the remainder of the siliceous selected ore, in a blast-furnace, to form a 55 per cent. matte for the converters, and a moderately basic slag, to go back to the ore cupolas.

3. Bessemering the matte from No. 2 up to high blistered copper in converters, the ferruginous slag therefrom resulting going back to the ore cupolas.

The first calculation required is to find out the exact amounts of the substances that we have to smelt. The proportion of pyrite, chalcopyrite, and gangue contained in the siliceous ore and in the concentrates has already been carefully established. The two pyritic minerals are known to be practically pure, and the average analysis of the slaty gangue has been settled.

COMPOSITION OF SILICEOUS SELECTED ORE.

80 tons (160,000 pounds) 10 per cent. Cu.

Copper pyrites	29.4	per cent.	Copper 34	per cent.
			Iron 31	"
			Sulphur 35	"
Iron pyrites	22.6	"	Iron 47	"
			Sulphur 53	"
Gangue rock	48.0	"	Silica 80	"
			Earths 20	"
	100.00	"		

ANALYSIS OF SILICEOUS SELECTED ORE.

(Deduced from above table.)

Copper			10.0	per cent
Iron	In 29.4 per cent. copper pyrites	— 9.12 per cent.		
	In 22.6 " iron pyrites	— 10.60 "	19.7	"
Sulphur	In 29.4 " copper pyrites	— 10.29 "		
	In 22.6 " iron pyrites	— 11.98 "	22.3	"
Silica in 48 per cent. gangue rock at 80 per cent.			38.4	"
Earths in 48 " " " 20 "			9.6	"
			100.0	"

COMPOSITION OF CONCENTRATES.

109.5 tons (219,000 pounds) 10 per cent. Cu.

Copper pyrites.......	29.4	per cent.	Copper..... 34	per cent.
			Iron........ 31	"
			Sulphur.... 35	"
Iron pyrites..........	60.6	"	Iron........ 47	"
			Sulphur.... 53	"
Gangue rock	10.0	"	Silica....... 70	"
			Earths..... 30	"
	100.00	"		

(There being some hornblende in the ore, the gangue is more basic in the concentrates than in the unwashed ore.)

ANALYSIS OF CONCENTRATES.

(Deduced from above table.)

Copper..				10.0	per cent.
Iron......	In 29.4 per cent.	copper pyrites	= 9.12 per cent.		
	In 60.6 "	iron pyrites	= 28.48 "		
			....	37.6	"
Sulphur..	In 29.4 "	copper pyrites	= 10.29 "		
	In 60.6 "	iron pyrites	= 32.11 "		
			....	42.4	"
Silica in 10 per cent. gangue rock at 70 per cent.......				7.0	"
Earths in 10 " " " at 30 "				3.0	"
				100.0	"

Of course these deductive analyses have been checked by many actual analyses of large lots of ores and concentrates from various portions of the veins.

Ore Cupolas.—The siliceous ore and concentrates just described have now to be melted raw in blast furnaces, with coke, for a 30 per cent. matte, using the slags from the mattte-concentration cupola and from the converters as flux. I will not attempt to give the reasons for adopting this somewhat peculiar method, whereby there is to be a concentration of only three into one in the first smelting. Yet they are very simple, when it is understood that coke and labor are excessively cheap, basic flux scarce, and that strong reasons exist for avoiding a calcining plant.

By using large furnaces, a great volume of blast, and slow running, there will be no difficulty in producing a 30 per cent. (or much higher) matte at the first smelting, and the heat produced by the combustion of the raw pyrites in the furnace will doubtless bring the coke consumption somewhat below the estimated amount, 10 per cent.

Owing to the complications introduced by smelting a portion of the siliceous selected ore with the matte in the second operation, and also by returning the ferruginous slags from that operation, and from the converters, to the ore cupola, we cannot calculate the ore-mixture as a straightforward proposition, but must begin by making some reasonable assumption, in order to get at the amount of slags that we shall have to resmelt. The slag from the ore cupolas should not carry over 0.4 per cent. copper at the outside, and, with the style of settlers provided, will not make over one-fourth of one per cent. of foul slag. This is so small an amount to be resmelted that we may neglect it entirely in our calculation; nor need we take into account the copper in the slags that are resmelted from the two last operations, as it is a constant amount, and is eventually recovered.

We will start our calculation for the ore cupolas with the following mixture:

per cent 10.0 3.3 5.3 18.1 35.3

	Pounds.	Copper	Iron.	Earths.	Silica.	Sulphur.
Siliceous selected ore........	120,000 with	12,000	23,640	11,520	43,080	26,760
Concentrates................	219,000 with	21,900	82,344	6,570	15,330	92,856
Total..................	339,000 with	33,900	105,984	18,090	61,410	119,616

To produce a normal 30 per cent. matte with the above quantity of copper, will use up sulphur and iron as follows: 33,900 pounds copper will make 42,375 pounds subsulphide of copper (80 per cent. matte), or 33,900 pounds copper will make 113,000 pounds of 30 per cent. matte, containing subsulphide of copper, 42,375 pounds; sulphide of iron, 70,625 pounds; a total of 113,000 pounds.

The iron which is thus taken up into the matte in the shape of sulphide of iron amounts to 44,917 pounds. Deducting this iron that we have thus temporarily lost from the total amount of iron contained in the mixture, we have 105,984 minus 44,917 equal to 61,067 pounds of iron still available to flux the silica of the mixture. This iron when oxidized to ferrous oxide, so that it can enter the slag, will weigh 78,471 pounds.

We also have a considerable amount of earths available to flux the silica, and as they consist almost exclusively of magnesia and a little alumina, we may call them worth twice as much, pound for pound, as the ferrous oxide, their lesser atomic weights making

C = 102 204
O = 55 110
O = 39 78
C = 71. 142
O = 81.4 162.8
gO = 40.3 80.6

largely in our favor. Therefore, we will multiply their weight by 2, and reckon them as ferrous oxide. We have then as available ferrous oxide:

61,067 pounds iron in charge................	— 78,471	pounds ferrous oxide.	
18,090 pounds earthy bases × 2...	— 36,180	"	"
Total available ferrous oxide.............	114,651	"	

This, with the 61,410 pounds silica in the ore, will give a slag containing 34.9 per cent. silica. This is somewhat more siliceous than we desire, nor have we left any leeway, which is desirable where siliceous ores abound, as it is always extremely easy to make the slag more siliceous if required, it only being necessary to select the ore a little more thoroughly, and throw a trifle less work on the concentrator.

We will here leave the ore cupolas temporarily, and take up the matte concentration and converter processes, which will show us how much, and what quality of slag we shall have to return to the ore cupola.

Matte Concentration.—In this operation we shall use a hot blast, and shall depend for our fuel mainly upon the combustion of the sulphur and iron in the matte and added ore. Whatever success may have attended pyritic smelting in general, no one who has had any experience in the matter denies the ease with which *matte* may be thus concentrated, providing that the blast is heated to 800 degrees to 1,000 degrees Fahr. (538 degrees C.), that the furnace is of the proper size and shape to ensure sufficient oxidation, that the blast is low in pressure and large in volume, and the smelting is intelligently managed. The very small percentage of coke that may be used to keep things in a comfortable condition (1 per cent. to 3 per cent.) will not make ash enough to demand consideration.

The composition of the charge will be as follows:

	Pounds.	Copper.	Iron.	Earths.	Silica.	Sulphur.
Matte from ore cupola.......	113,000 with	33,900	44,917	....	....	34,183
Rest of siliceous ore.........	40,000 with	4,000	7,880	3,840	15,360	8,920
Total....................	153,000 with	37,900	52,797	3,840	15,360	43,103

37,900 pounds copper will make 68,910 pounds of 55 per cent. matte, consisting of

Subsulphide of copper	47,375	pounds.
Sulphide of iron	21,535	"
Total	68,905	"

The iron thus taken up in the matte as ferrous sulphide amounts to 13,696 pounds, which, when deducted from the total amount of iron that was in the charge, (52,797 pounds), leaves available for slag formation

Iron	39,101 =	50,245	pounds.
Earthy bases, reduced to value of ferrous oxide	3,840 × 2 =	7,680	"
Total ferrous oxide		57,925	"

As there are only 15,360 pounds of silica in this charge, the above amount of ferrous oxide would make a slag containing only about 21 per cent. silica, which is much too low for good work. As we desire in this furnace to form a slag containing about 28 per cent. silica, we require to add to the charge about 7,166 pounds silica. It is well to thus leave a point where we are sure of a little basic excess, for where siliceous ores abound, the ore-cupola slag always seems to run a little more acid than we anticipate, and it is easy enough to cancel this margin whenever desired, either by using a few more tons of unconcentrated ore as already suggested, or by adding a daily proportion of the rich, siliceous slimes from the concentrator, which are invariably only too plentiful under the assumed conditions.

In order to avoid too great length of calculations, I will assume that the required silica is added as pure, non-cupriferous quartz, although of course this would not be done in actual work.

Then the total weight of slag to return from this process to the ore cupolas will be as follows:

Ferrous oxide	57,925	pounds.
Silica in charge	15,360	"
Added silica	7,166	"
Total	80,451*	"

The converter plant will be required to take care of the 68,910

* The weight of the matte-concentration-cupola slag may be taken at the above figure in estimating its chemical effect in the ore cupolas, but its actual weight is a trifle less, owing to the doubling of the weight of the earthy bases, to make them equal in effect to ferrous oxide. This will be corrected later.

pounds of 55 per cent. matte from the matte-concentration cupola. This matte has the following composition:

Copper	37,900	pounds.
Iron	13,696	"
Sulphur	17,314	"
Total	68,910	"

The iron in the above matte, 13,696 pounds	= 17,600	pounds FeO.
To make a slag of 30 per cent. silica it will require	7,543	" SiO_2.
Total	25,143	"

(The small amount of alumina taken up in this slag from the clay of the converter linings may be disregarded, both because its proportion to the total amount of material smelted in the ore furnaces is almost infinitesimal, and also, because, in such a ferruginous slag, its presence is rather welcome than otherwise, as tending, in some slight measure, to decrease its specific gravity.)

Complete Ore Cupola Charge.—We now have the data from which to calculate the total quantity of material that will come to the ore cupolas, and can thus estimate the quantity of coke required, and allow for the ash in the same. The coke consumption will be very low. Much of the charge consists of sulphides, and considerable heat will be generated by the combustion of their sulphur and iron. At the Butte & Boston smelter at Butte, Montana, Mr. Allen is smelting raw sulphides in a cupola with 10 per cent. of coke, and this without any especial attempt to profit by the heat resulting from their oxidation. In the case under consideration, where the furnace will have great area, perpendicular walls, and low pressure with large volume of blast, the pyritic effect will be considerable, and both the ratio of concentration and the consumption of carbonaceous fuel will be benefited thereby. But, for conservative reasons, I will estimate the consumption of coke at 10 per cent. on the entire charge, or about 13 per cent. on the weight of the ore smelted.

About 44,000 pounds of coke will be used, containing some 10 per cent. ash, having the following composition:

Silica	56	per cent.
Ferrous oxide	21	"
Earthy bases	23	"
Total	100	"

TOTAL MATERIAL TO ORE CUPOLAS (SULPHUR AND OXYGEN OMITTED).

	Pounds.	Copper.	Iron.	Earths.	Silica
Concentrates..................	219,000 with	21,900	83,344	6,570	15,330
Selected siliceous ore..........	120,000 with	12,000	23,640	11,520	46,080
Slag from matte concentration.	80,451 with	neg.	39,101	3,840	22,526
Slag from converters...........	25,143 with	neg.	13,696	neg.	7,548
Coke ash.......................	4,195 with	....	719	1,012	2,464
Total...........	444,594 with	33,900	159,500	22,942	93,943

The above table gives simply the slag-forming constituents of the ore-cupola charge and the copper, omitting part of the oxygen brought into the charge by the slags from matte-cupola and converters, and also omitting the sulphur, which possesses no interest for us in the calculation, and has already been given in the preliminary figures. We can be very certain that there will be no difficulty in inducing the copper and iron to take up enough sulphur to form a 55 per cent. matte, and all sulphur beyond this must be burned to sulphurous acid gas. This can be done with ease and certainty, and herein lies the main *chemical* difference between the old and the most modern practice.

After deducting the 44,917 pounds iron, which, as we found before, is temporarily lost in forming the 30 per cent. matte, we have the following slag-forming materials remaining in the mixture:

Silica..	93,943	pounds.
Earths, reckoned as ferrous oxide...............	45,884	"
Iron, as ferrous oxide	147,240	"
Total......................................	287,067	"

This gives us a slag consisting of silica.........	32.72	per cent.
Ferrous oxide (or its equivalent)...............	67.28	"
	100.00	"

This gives us our desired slag of about 33 per cent. silica, with such proportion of earthy bases as the ore and coke-ash will afford. Run slowly, and with "reverberatory-settler" (to be described later), this slag need not contain over 0.4 per cent. copper.

The *actual* weight of the slag may be determined by deducting the amount of the earthy bases from the total weight of the ore-cupola slag already given (287,067 pounds). This is necessary because the weight of the earthy bases was doubled, in order to make them equal in chemical effect to ferrous oxide; 287,067 minus 22,942 equals 264,125 pounds, which will be the total weight of the slag produced in the ore cupolas every 24 hours.

This is as far as it is suitable to carry the illustration, as it is not intended, at this point, to describe the planning of the works to treat the above ore. I will only add that to smelt the 222 tons of ore and slag that are to come daily to the ore cupolas, I should use three large blast-furnaces, thus giving each a duty of only 74 tons of charge, or 57 tons of ore. This low duty will permit of the slow running essential when the blast-furnace is to be used as an oxidizer and partial generator of its own heat, and will also permit ample stoppages for repairs without any diminution of the output. It will scarcely add anything to the cost of smelting per ton, as the charging will be done with mechanical aid, and there will be one weighman whether there are two or three cupolas. As the slag is to be granulated and removed by water, the item of pot-haulers does not enter into consideration, and the moderate and comfortable running done by three furnaces, as compared with trying to smelt the same amount of ore in two cupolas, will save enough work to supply the labor needed below at the third furnace.

The matte-concentration will require only one cupola, with hot blast. There are only about 80 tons of material to treat daily, and it will be more difficult to "hold back" the matte, than it will to put it through. It may be necessary to run a more siliceous slag at this cupola to prevent too rapid smelting, matte being so heavy and so fusible that it is difficult to restrain it long enough to gain the oxidation necessary for its proper concentration. In the present case there will be no trouble, however, as the concentration aimed at is less than 2 into 1. Any extra desired acidity of the slag will be a welcome circumstance to the concentrator foreman, as relieving the pressure in the slime department.

So little flue-dust will be made with the light blast and slow running, and the capacity of the furnaces is so ample, that it need not be here considered.

CHAPTER XI.

BLAST-FURNACE SMELTING (WITH CARBONACEOUS FUEL).

THE principal developments in the American system of blast-furnace practice had already long taken place at the time of the publication of the first edition of this work. The improvements since that time have been characterized by perfecting of details, a simplification and economy in the method of manipulating the furnace and its accessory apparatus, and a decided saving in the handling of charge and product, rather than by any radical change of principles.

I do not hesitate to call it the *American* system of blast-furnace practice; for its advance on the German process whence it sprang is so marked, and its whole style of working so radically different, as to constitute a new departure.

Twenty-five years ago the copper blast-furnace was regarded as an intricate, eccentric, and highly uncertain machine, erected on deep and massive foundations, enclosed in spacious and expensive buildings, and provided with one to five tuyeres of limited area, through which a gentle stream of air trickled into the interior, without disturbing their most important feature, the "nose."

The tamping-in the bottom of this furnace and its long, brasque forehearth, and its subsequent careful drying, was a ceremony that lasted days, and led up to that culmination of the metallurgist's skill and responsibility, the "blowing-in." Every charge was then watched as it descended, and the subtraction of half a scoop of coke, or the substitution of a shovel of ore for a similar amount of slag, were matters for grave consideration and argument. Even after a day or two when the furnace was in full blast, it was generally thought necessary to use from 20 per cent. to 50 per cent. of slag in the charge; and, indeed, owing to the imperfect settling of the matte, there was seldom any lack of foul slag for the purpose.

The charging was done with the utmost care and on the most minute scale, a charge often consisting of but 200 pounds, which was painfully distributed around the walls of the shaft; and the smelting, together with the influence of the finely-broken and thin layers of ore, caused such a powerful reducing action, iron "sows" were a constant menace and frequent reality. leed, certain smelting works were provided with special furnaces, here these metallic masses were subjected to a "Verblasen," or ification, to recover what value they might contain.

The campaigns were short, and, like nations, were characterized by a long period of very gradual rise, a short interval of maximum prosperity, and a protracted and most painful term of decadence and waning productiveness.

Fifteen to 30 tons of ore per 24 hours was considered a fair duty for a copper furnace, and the campaigns seldom lasted for more than a month.*

The present American copper cupola of the most advanced type consists of a circular, or oval, water-jacketed shell—the inner skin sometimes being of thick sheet-copper, to withstand the corrosive action of damp ores that contain sulphates (quenched calcines)—or of four or more straight wrought jackets, that are clamped together to form the sides and ends of a rectangle, perhaps 40 by 160 inches. The tuyeres are ten to twenty in number, contrived so that their diameter may be varied, and arranged so that the blast in each one may be independently controlled. The blast is derived from a positive, or semi-positive blower, and furnishes at least 7,000 cubic feet of air per minute, at a pressure of two inches mercury (690 mm. water). The blast never ceases, except in case of accident or repairs.

The molten products escape at once from the brick bottom of the furnace into a brick-lined movable forehearth, of large dimensions. From this, the thoroughly settled slag flows in a constant stream into large pots drawn by mules, or into a stream of water, which granulates and removes it. In some large works, the matte is tapped in charges of five tons into clay-lined ladles moved by an electric crane, which pours it immediately into a Bessemer converter, where it is blown up to 99 per cent. copper in a single operation, and cast direct into anode-plates for electrolytic treatment, if it contains the precious metals; otherwise, into pigs for the refinery.

The amount of foul slag to be resmelted need seldom reach one

* The Mansfeld practice has always been exceptional, owing to its unique conditions.

per cent., and the substitution of a new forehearth every few weeks is the only ordinary delay; and this, a very brief one. The operations of blowing-in and blowing-out are regarded about as seriously as they would be at a foundry-cupola. In blowing-in, the foreman usually begins with a few slag-charges, and after a few light charges of ore the furnace is in its normal working condition. The charging is done directly from cars or large barrows, and the ore charge for a furnace of this size would be about two tons. The length of the campaign depends upon the durability of the water-jackets and machinery, and the prevalence of strikes.

In a word, the American copper metallurgist regards a blast-furnace as a simple cavity, surrounded by a fireproof wall, in which his mission is, to burn coke with the greatest attainable rapidity, taking care always to supply the utmost quantity of carefully fluxed ore that the coke can melt, and forcing his charge through the furnace so quickly that there is no opportunity for the reduction of iron to a metal; while the instantaneous removal of all molten material still further prevents the formation of metallic iron, enables the products of fusion to settle quietly and thoroughly according to their weight, and removes the great source of troubles, delays, and repairs from the inside to the outside of the furnace. A daily duty of 100 to 160 tons of ore is attained, and from late experiments with ample blast and not too fine ores, I have little doubt that we shall find it economical to use furnaces with a daily capacity of some 300 tons of ore.

The granulation of the slag by water, and the use of furnaces with the gases drawn off below the charging-floor, so that the tunnel-head remains open and unobstructed, is in common use, and permits the use of an automatic car, the entire length of the furnace, which will drop its charge instantaneously. (Pueblo Silver-Lead Smelter.) This will remove any difficulty that might be encountered in attempting to handle so great a quantity of material at a single furnace in works not suitably constructed therefor.

The practice of blast-furnace smelting in the United States almost invariably implies the employment of a water-jacketed, or water-cooled, furnace. Even the large brick Raschette furnaces, so skillfully managed and so firmly adhered to by the Orford Copper Company, have been cooled for many years past by pipes buried in the brick-work through which water circulates.

With so many skilled and thoughtful engineers and foremen in charge of our copper plants, and in the face of the grinding economies that have necessarily accompanied the marked decrease in the price of copper and silver, it is not probable that the water-jacket would be so universally employed, did it not possess decided economies and advantages as compared with its unprotected prototype. Any reasonable suggestion or innovation obtains patient hearing and prompt trial in this country, and no pattern of brick furnace that offers any encouragement for cheaper work would have, or has had, long to wait before being somewhere given an opportunity to prove its claims.

During eight years of metallurgical work I used nothing but uncooled brick furnaces, with, or without, water tuyeres, and I think that a brief comparison of general results with subsequent water-jacket practice may be of interest. I feel the more satisfied of the correctness of these views from finding that I hold them in common with all American metallurgists with whom I have conversed on the subject, and whose experience comprises both classes of furnace.

Where passable water is obtainable at any reasonable expense, the first cost of the two types of furnace is pretty nearly the same. A large, sectional, copper-lined water-jacket of the most modern type, with deflecting tuyeres and independent tuyere-valves will cost considerably more than a simple, lightly built brick furnace. On the other hand, a massive, thick-walled brick furnace with appropriate foundation will cost more than a plain, but perfectly good and durable jacket of the Bartlett type. And in any case, the difference in first cost is but a trifling matter compared with even the slightest degree of efficiency or economy of one furnace over the other. We may assume, therefore, the cost of the two furnaces to be equal. The main advantages claimed for the water-jacket type are:

1. *The Ease with which it is Planned, Constructed, and Erected.*—It can be planned at leisure, and the working drawings sent to the place where it is to be made. Then, after digging a hole and preparing a block of concrete, masonry, or slag to set it on, the subject can be dismissed from one's mind until the furnace arrives complete and ready to set up.

It can be erected by the most ordinary mechanics and in a very few days. This is a great relief to the metallurgist accustomed to constructing brick furnaces, with their various items of fire-brick, red brick, mortar, clay, buckstaves, tie-rods, arch-patterns, etc. I have had a water-jacket furnace running steadily on the third day after the wagons containing it had arrived.

2. *The Simplicity of the Blowing-in Process.*—We have learned to make less and less of this once awe-inspiring operation, yet even now the blowing-in of a large brick furnace is a slightly precarious task. The least excess of fuel or pressure of blast is likely to cause very serious damage to the new brick-work, while an atom too low a temperature is certain to start accretions in the hearth and about the tuyeres, and too light a blast may leave a raw core of ore in the middle of the shaft that is sure to cause much trouble and delay.

This is especially the case when a new hearth or bottom has been constructed, the proper drying and warming of the same demanding some 24 hours. The heating up of the great mass of brick-work forming the shaft is also a slow operation, and absorbs a vast amount of heat for the first twelve hours or more.

But all this is but a small matter compared with the burning out of the hearth and walls. With the fast driving, abundant coke, and basic charge usually employed in starting a new furnace in this country, it seems at times impossible to maintain a perfectly uniform condition in the furnace shaft, and some one corner or other is extremely likely to begin burning out, and to defy every effort to stop it, until the brick-work is thinned enough to feel the cooling influence of the external air.

3. *Ease and Cheapness of Repairs.*—This was once a disputed point between the adherents of the two types of furnace. At present, the water-jacket men can find scarcely any one to dispute with. The few brick furnaces that are run in this country are managed with the greatest care and skill, and every precaution and manœuvre that years of experience can suggest is brought to bear upon them. With water-jackets the case is often the reverse. These furnaces are constantly started at new mines with inexperienced men, and with mismanagement and abuses that are scarcely credible. The common impression seems to be that nothing can damage a water-cooled furnace, and that so long as it does not show symptoms of chilling, and the slag does not carry too much metal, everything is right. A very small fraction of the carelessness that is so frequently displayed in running a water-jacket would ruin a brick furnace within twelve hours. It is sometimes a question whether this extraordinary capacity to withstand too much fuel, or an improper slag or matte, is not a positive disad-

vantage, as encouraging waste and carelessness. Consequently, to arrive at anything like a fair comparison of the cost of repairs, we must consider the two furnaces under the same conditions, so far as is possible. And, as a brick furnace cannot be profitably run (in the rapid manner common to this country) at all, without skilled supervision and thoroughly experienced foremen and workmen, we can only compare it with a water-jacket run under equally good management. This greatly lessens the apparent advantage of the water-jacket, as it excludes all cases of careless management, under which it is probable that the discrepancy in the cost of repairs and renewals would be multiplied many times over.

Under the favorable conditions referred to for both furnaces, I estimate that the cost of repairs on the brick furnace, and the proportion of sinking fund to renew it when worn out, amount to something over double as much as with the water-jacket.

The comparative loss of time from delays shows even more unfavorably for the brick furnace. These points will be considered in detail when we come to treat of the expense of running.

4. *Convenience of Manipulation.*—The advantages here are all in favor of the protected furnace. With the volume and pressure of blast necessary to put 100 tons or more of ore through a furnace every 24 hours, and with the ordinary necessity, or desirability, of running a continuous stream of slag, there is a strong tendency for a portion of the blast to escape through the slag-hole, carrying with it a stream of glowing cinders and chilled slag and matte-globules, and often causing a heavy loss in values on argentiferous ores, especially if a little lead, zinc, or antimony be present. The loss in heat and pressure are also considerable, and the workmen are annoyed by the heat and noise of the escaping flame. In water-jackets, even where there are none of the ordinary water-cooled tymps, or trapping devices, the escaping blast is suppressed with comparative ease, so long as there exists a cold and unattackable border to the slag-hole, against which brick and clay can be solidly built. The weak point of this outside dam is its junction with the front wall of the furnace. In water-jacketed furnaces, as the heated clay shrinks away, the resulting crevice is quickly sealed by the slag that bubbles out with the blast, and a skilled furnace-man will make even such a rude defense as this last for several hours. With brick furnaces, however, the crack widens rapidly as the sharp edges of the brick work are melted away by the blowpipe action of the flame. More clay is piled on in great balls until the

front of the furnace is provided with an excrescence resembling a small haycock, and the buried brick front losing its only chance of being cooled (external radiation), softens and melts away more or less completely, requiring the entire removal of all the débris, and the rebuilding of the front. I need hardly say, that in all well-arranged water-jackets, the blast is so trapped as to avoid even the mild form of loss and annoyance arising from the cause just mentioned.

Another highly important advantage possessed by water-cooled furnaces is the ease with which accretions are removed from the walls of the shaft. Nearly all ores of copper contain a little zinc or lead, and a thick coating of these metals soon settles on the walls of a furnace. This deposit becomes so extensive in the upper portions of the shaft that it would greatly lessen the capacity of the furnace and also alter its reducing (or oxidizing) power to an extent that would seriously affect the composition of the matte and slag, were it not barred off from the tunnel head at regular intervals of a few days or weeks. The ore-charge having to be allowed to sink as far as practicable, the red-hot walls of a brick furnace make this barring process a most prolonged and painful task, not only on account of the excessive heat, but also because the volatilized sulphides soak into the softened brick-work until they have to be actually chiseled away at every point. In the water-jacket, on the contrary, when the ore has sunk below the accretions, the furnace shaft is comparatively cold, and, after a charge of cold coke and ore has been thrown upon the glowing mass below, it is by no means an arduous task to bar away the crusts from the furnace walls, especially as their adherence to the cooled iron is very slight, and when a small portion of the ring is once chiseled away, the entire mass usually falls to pieces.

In the water-jacket furnace, the operation of blowing out is also bereft of most of its heat and toil, and is so slight an affair that, after the charge has sunk nearly to the tuyeres, the furnace can be tapped dry, the forehearth removed, the loose coke and cinders still remaining dragged out and quenched, all within an hour, and a workman can immediately enter the furnace if repairs are necessary, a few inches of ashes being thrown onto the hearth to protect the board that he stands on. In 1½ hours from the time he is through, slag can be running again at pretty nearly the normal rate.

The above are a few of the more striking advantages offered by the water-jacket furnace, but there is a very much longer list of lesser advantages that will be noticed in describing the management of blast-furnaces, and that form, when assembled, an overwhelming argument in favor of the water-jacket cooled apparatus.

I know of but three reasonable arguments that are commonly advanced against the employment of the water-jacket. These are:

1. The scarcity and impurity of water in certain localities. If there is absolutely insufficient water, it is evident that a water-jacket furnace cannot be used. But wherever water can be obtained at any reasonable trouble or cost, it is equally certain that it will pay to do it. The impurity of water has been the cause of considerable annoyance at certain smelters in times past, but much has been done in the way of improved settling arrangements for both mechanical and chemical impurities, and water-jackets are now run steadily at places where formerly there were many delays and much expense from this cause. Experience has also taught us how to construct the jackets to suit them to such conditions, and it must now be a very foul water that is not preferable to no water at all.

2. The danger of ruining the furnace by careless management of the feed water. This is a very curious objection, and applies with much greater force to steam boilers or to water-tuyeres or coils. For any overheating of the jacket-water is immediately shown by the puffing and steaming of the discharge pipe, and it is astonishing how difficult it is to seriously damage one of these furnaces, even when there has been the most criminal carelessness and the jacket has been allowed to boil away half its water contents. The dangerous temperature is all in the neighborhood of the tuyeres, and long before the water level has sunk to that point the furnace will have proclaimed its needs in a manner so unequivocal as to startle even a night foreman. A mere stoppage of the blast is sufficient to restore matters to comparative safety while the fault is being repaired. And, lastly, if the furnace-men are so abnormally irresponsible and unintelligent as to make it possible that such a condition of affairs should occur, it is perfectly easy to arrange an alarm bell so that it will act like the danger signals on the railroads, remaining quiet while everything is in proper condition, and sounding a shrill and continuous alarm as soon as the jacket-water rises above a certain maximum temperature. This is effected by an electrical connection with a plug of fusible metal in

the water space, which will melt at say 180 degrees Fahr. (82 degrees C.).

3. That the water-jacket wastes fuel seriously in heating the cooling-water. This is a grave charge, as much of the blast-furnace smelting in America is done with coke at $12 to $15, and even $40 per ton. To have any clear opinion on this question, apart from general knowledge derived from practice and comparison, it is essential to first determine how much fuel is actually required to heat the water used in cooling a jacket furnace run at the rapid rate now generally adopted. Mr. H. M. Howe, in Bulletin No. 26 of the United States Geological Survey, gives some figures on this subject, made by Mr. J. B. F. Herreshoff, of The Laurel Hill Chemical Works, in 1884. Mr. Herreshoff is such a competent and careful observer as to make his figures of particular value.

The furnace was a round, wrought-iron water-jacket with 2-inch water space, the jacket extending from bottom of hearth to charging door, and thus exposing an unusual area to the heat. It was 52 inches in diameter at the tuyeres and 10 feet high, having ten 2-inch tuyeres. It averaged 90 tons (180,000 pounds) per 24 hours of roasted 6 per cent. pyrites, with a consumption of 12 tons of gas coke, making a 45 per cent. matte, and a slag with 31 per cent. silica, 52 per cent. ferrous oxide, and 0.55 per cent. copper.

HEAT ABSTRACTED BY JACKET-WATER.

Initial temperature of water	15.5 degrees C.
Final temperature	77
Gallons water per hour	2,000
Pounds of coke required per 24 hours to heat jacket-water, assuming a useful effect of 25 per cent. of the calorific power of coke	1,328
Pounds coke for jacket-water per ton ore smelted	14.7
Value of this coke per ton ore smelted, at $5 per ton coke	$0.039
Percentage of total coke consumption used in heating jacket-water	5.5

I have made a number of similar tests on different furnaces running on various classes of ore, and under widely diverse conditions —although always with large capacity. The results vary very considerably according to the state that the furnace happens to be in on the day of the test, and especially according to the physical condition of the ore—whether fine or coarse, porous or massive, wet or dry, etc. They are also greatly influenced by the capacity of the ore to form a coating of lead or zinc sulphides on the inner surface of the jacket, which decidedly lessens the loss of heat to the water. Such a coating is highly advantageous if it does not grow too rapidly, and it is preferable not to have the rivet-heads too flat or countersunk on the interior of the shell, as they give just the slight support required to prevent this useful crust from falling off at intervals into the furnace and creating irregularities, as well as increasing the consumption of fuel. In the various tests referred to, I have found that the coke wasted in heating the jacket-water varied from 2½ per cent. to 10½ per cent. of the total amount used. I am inclined to think that about 6 per cent. is the maximum allowable figure under normal conditions and that if much more than this proportion is being used, one of three things is happening: Either

1. Too much coke is being charged, or

2. The method of charging is wrong, and too much coke is being consumed in contact with the furnace walls, thus wasting a considerable proportion of its effect, or

3. The circulation of the water in the jacket is too rapid, and the water is escaping too cold.

Mr. Howe's table shows that the Laurel Hill furnace is expending about 4 cents per ton of ore smelted, in heating its jacket-water. On 90 tons burden per day, this is at the rate of 15 cents per hour.

Now it is practically impossible to determine the average loss of heat by radiation from the walls of a brick furnace, nor is there the slightest sense or object in comparing this factor with the heat used in warming the jacket-water. If we consider the question of radiation at all, we must compare the loss by radiation from the inner surface of the brick furnace with the loss by radiation from the outer surface of the water-jacket, which, as I need scarcely point out, is largely in favor of the latter. When we come to consider the fuel wasted in heating the jacket-water, we can only compare it with the damage done to the brick furnace-walls, and the heat wasted in raising them beyond a proper temperature. We cannot separate the damage (and waste of fuel) occasioned by the heat, and that done by the fluxing action of the ores on the fire-brick. But it is not in the least necessary to make this separation, as both these sources of expense are avoided by water cooling, and, consequently, both must be counted against the brick furnace in making our comparison.*

Therefore, if any metallurgist is not content to pay 15 cents per hour (at New York prices) to guarantee the perfect integrity of his furnace walls and breast, he is either more skillful, or more ignorant, than most copper men in this country.

Commercial results and the general testimony of skilled, practical metallurgists are, after all, more reliable than the imperfect tests and comparisons that we can make on this point. So far as my knowledge extends, these are practically unanimous in regarding the water-jacketed furnace as the most convenient and economical pattern of blast furnace for copper or lead ores.

This being the type of furnace used almost exclusively in this country, all general remarks on blast-furnace smelting may be considered to apply especially to water-jackets. A special section will be devoted to the consideration of brick blast-furnaces.

WATER-JACKET BLAST-FURNACES.

These may be divided into two classes, according to the material of which they are constructed:

1. Jackets made of cast iron.
2. Jackets made of wrought iron, soft steel, or rolled copper.

1. *Cast-iron jackets* are necessarily built in sections, the various jackets being assembled and clamped together to form the complete shaft. The lead-silver smelters have been mainly instrumental in introducing this type of furnace into the domain of copper metallurgy. Having found it to answer admirably for the quiet, moderate, and regular furnace work characteristic of the well-conducted lead-smelting process, they have naturally carried this furnace along with them, as the diminution of rich lead ores and the transition in depth from oxide to sulphide ores have forced them into matte smelting. The rapid driving and, at times, fierce overheat of the copper furnace, accompanied by frequent irregularities resulting therefrom, and from the less careful fluxing of the ores, make the cast-iron jacket inconvenient for the matte smelter. I am aware that many excellent metallurgists differ from me in this opinion, but I have run both types of furnace under many differing condi-

* It will be understood that nearly all these remarks apply to the conditions that prevail in America, where furnaces are usually run at high pressure, smelting from 75 to 150 tons per day.

tions, and, with all reasonable care and attention, I have found the delays arising from the occasional cracking and replacing of a jacket to greatly exceed any possible increased first cost of the wrought-iron furnace. Cast jackets are especially liable to crack in cold climates and during the operation of blowing-in; and, as it is frequently necessary for the copper smelter to start up a furnace for a short campaign, it is of prime importance for him to have it capable of withstanding all the fluctuations of temperature that may occur under such circumstances. I know of no possible conditions under which I should not choose the wrought-iron furnace.

Cast-iron water-jackets are so thoroughly illustrated and described in our modern text-books on lead-silver smelting, that it would be contrary to the design of this work to repeat this information. I will only mention a few practical points that are of especial importance to the copper smelter.

1. It is important to obtain jacket-castings from a foundry that has had considerable experience in making them, and has made a study of the mixture of irons best suited to them.

2. A plan of construction should be adopted that will enable the various sections composing the furnace shaft to be keyed together, and unkeyed, with the greatest possible facility, despatch, and firmness. The bustle-pipe, tuyere branches, and feed-water connections should be as few and simple as possible in their arrangements, and so planned that they can be taken down or put up in a few moments. In this way, the delay resulting from having to change a jacket during a campaign will be reduced to a minimum.

3. There should be as few different patterns of jackets as practicable in the furnace; otherwise, too many castings must be kept in stock.

4. It prevents cracking and saves fuel to run the jacket-water pretty hot, say 160 degrees Fahr. (71 degrees C.), or more. The temperature should be kept as uniform as possible and in the various jackets, though when it is decided to establish a continuous circulation through all the jackets by connecting them externally with 2½-inch U-shaped bends, it greatly assists the circulation to run the feed-water into the two end-jackets, and keep these 20 degrees or 30 degrees cooler than the others.

The main difference between running a cast-iron and a wrought-iron jacket is, that with the former we have to be more careful in warming and blowing-in the furnace, and that we occasionally have the unpleasant and unprofitable task of changing a jacket while the furnace is in blast. It is a hot and disagreeable job at the best, but the foreman who takes the most time and pains to arrange all the preliminaries, and to make it cool and comfortable for his men, will usually be found to accomplish it more thoroughly and more quickly than the one who attempts to rush things and to work at arm's length over a mass of red-hot ore and glowing coke.

A leak in a water-jacket is not often so dangerous a catastrophe as it may seem. When it is between the edges of two adjoining jackets, the chilled slag on the inside will soon force the water to seek an external path, and when it is on the outside, it can easily be conducted away from the base or crucible. Even when the leak is internal it may not always be serious, as furnaces with an iron base-plate instead of a brick hearth will usually right themselves, the chilled slag and matte forming a dam between which and the walls of the furnace the water will usually be retained until it can seek an outlet, perhaps between the bottom-plate and lower edge of the jackets. But if the leak continues to be serious, and cannot be stopped by the use of oatmeal, horse dung, and other approved sediments introduced into the affected jacket, it is better to change the section at once; for not only is there more or less danger of an explosion, but the hearth is sure to be chilled and the flow of slag and metal obstructed by the steam generated inside the shaft. In a furnace with a brick base, the water may sometimes be drawn off from the interior by driving a heavy steel bar from the exterior to the probable locality of the self-occluded, interior pool, but it is a risky and temporary measure.

If the section of jacket is to be changed, the foreman should first make sure that everything that is needed in the operation is at hand and in readiness for immediate use. The charge being pretty low in the shaft, its glowing surface should be covered with a thick layer of dampened small coke and ashes, and two or three long bars should be driven down from above close against the inner surface of the leaky jacket, and until their points are in contact with the hearth. The air and water connections are then quickly broken and removed, and the fastenings of the condemned jacket are unkeyed or unbolted, jackscrews, if necessary, being placed against the adjoining sections to keep them from being forced out of position.

The red-hot ore and coke that escape through the gap are promptly dragged to one side and quenched with water. The glowing column of charge that is seen on removing the section of jacket is only prevented from escaping *en masse* by the bars driven from above; and opposite it, the heat is too great to permit of the rapid replacement of the new section. By means of a number of strips of heavy, refuse sheet iron, about 8 inches wide and somewhat longer than the breadth of the open panel, the latter is temporarily closed, the sheet-iron strips being handled with tongs and bars, and inserted into the opening so that they span it from side to side, their extremities catching inside the two adjoining jackets. They are strengthened by short iron rods that are also so inserted as to catch on the inner surface of the jackets. This barrier keeps the glowing charge in place, but scarcely diminishes the powerful radiation from the opening, as the sheet iron becomes red-hot in a moment. One hundred pounds, or more, of well-puddled, sticky clay is in readiness, and being thrown in large balls against the iron casing, flattens out and forms a thick coating impervious to the heat for five or ten minutes. In this time, the new jacket should be replaced, filled with warm water from a hose, and the wind and water connections made at once. A light blast can then be put on, and the furnace filled to its normal height. The sheet-iron and clay that are opposite the new section will soon disappear, forming, while they last, a good protection for the new jacket until it gets warmed up to its work.

The supporting of the jackets may be effected, either from an iron frame resting on columns, or they may be built up directly on the brick base of the furnace, or even on an immovable base-plate. The former method is the more customary and convenient, as it renders the hearth and shaft of the furnace entirely independent of each other. The principal manufacturers of furnaces in this country have various excellent designs that are the outcome of accumulated experience. Want of space forbids my going into these details that have been thoroughly worked out and established.

2. *Wrought-Iron Jackets.*—(Also made of soft steel, or rolled copper).—This is the most common and useful type of American copper blast-furnaces. The simplest and most general is the ordinary, circular wrought-iron jacket, extending in a single piece from below the tuyeres to a point well up the shaft, the total length being usually from 6 feet to 9 feet. The diameter usually increases toward the top at the rate of one inch, or more, per foot of vertical length. The tuyere openings consist of cast rings inserted into the water-space, a circle of rivets holding the inner and outer shells in close contact with these castings. The water-space

at top and bottom, and the slag opening, may be closed by wrought-iron rings to which the two shells are riveted, or the inner shell may be flanged over and riveted to the outer, forming a right angle to the vertical axis of the furnace. The bottom may consist of drop-doors like a foundry cupola, or of a simple cast-iron plate (dished, to prevent cracking), and bolted to the bottom ring of the

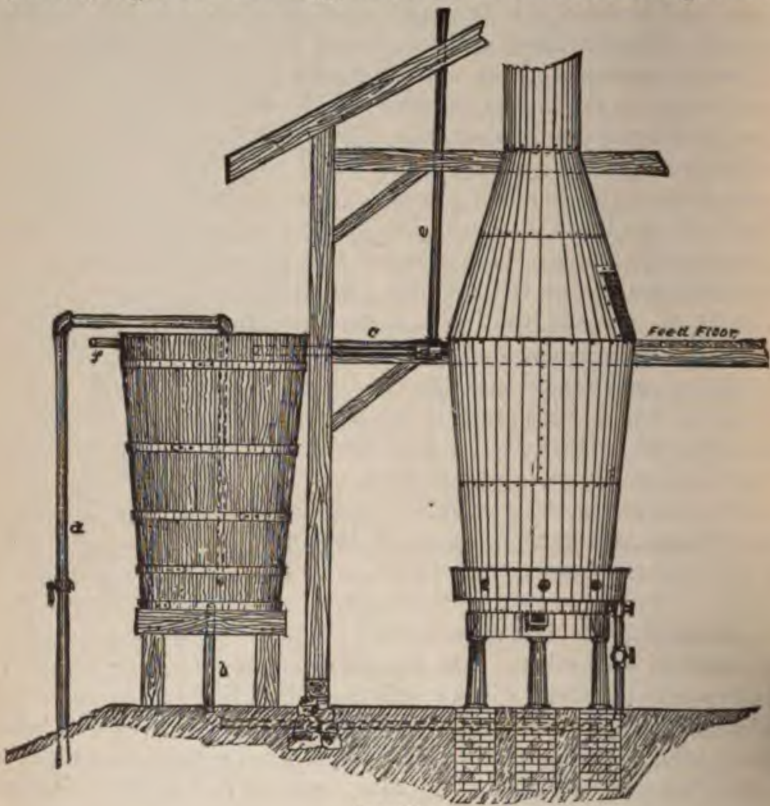


FIG. 26.

furnace; or it may be built up from the ground in the shape of a brick hearth of circular, oval, or rectangular form, securely braced by an iron shell. The accompanying illustration of Bartlett's water-jacket (Fig. 26) shows this simple and economical type of furnace. In large smelters, where greater capacity is required, the circular shape of furnace cannot be adopted, as a moderate blast will not penetrate a column of average charge to a greater depth than 20 inches to 24 inches. Consequently, a diameter of

48 inches or 50 inches at the tuyeres seems to me the extreme limit in this direction, even with a coarse charge. By allowing the tuyeres to penetrate the furnace shaft a few inches, the diame-

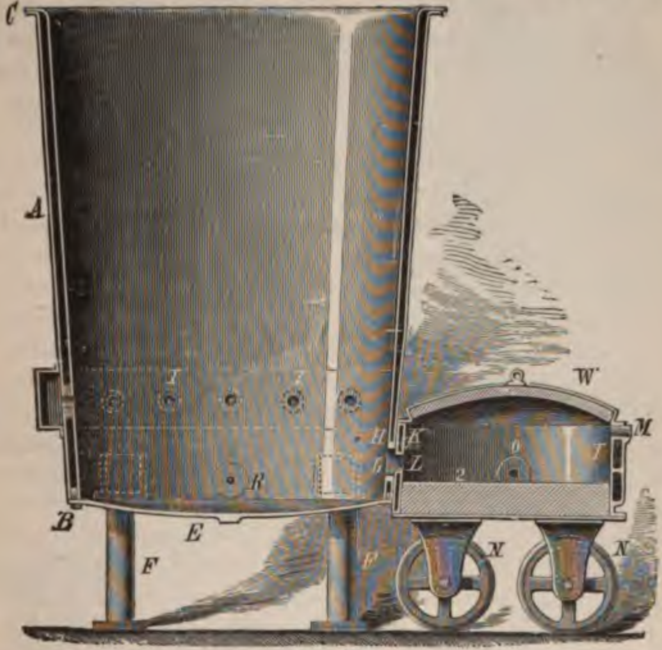


FIG. 27

FIGS. 28.

ter of a circular shaft may be increased to 54 inches, or even to 60 inches, with a nearly corresponding increase of capacity; but the complications resulting from the necessity of cooling these projecting tuyeres, and from other causes, have thus far outweighed the advantage gained.

Obviously our only recourse is to lengthen the furnace shaft in one direction, keeping it sufficiently narrow in the other dimension for the blast to penetrate to the center. This brings us at once to the rectangular form, or, if we desire to still make the entire jacket in one piece, we may construct it in the shape of a flattened oval. Mr. J. B. F. Herreshoff of the Laurel Hill Chemical Works, New York, has done this with much success, his improved furnace being shown in Figs. 27, 28.

*Figs. 29, 30, 31 and 32, are illustrations of water-jacket furnaces largely employed throughout the West in producing black copper from oxidized ores, or matte of tolerably high grade from sulphide ores. It is evident that the hearth would not stand very large amounts of low grade, fiery matte (below 30 per cent. copper); but, for the purposes intended, the cooling by radiation is generally sufficient to keep the bottom cool, though Walker has lately found it useful to use this radiated caloric in preheating the blast, at the same time keeping the hearth at a safer temperature. One to two thousand gallons of water per hour (3,785 to 7,570 liters) is needed to cool the jacket. The water is admitted through the pipes F, and escapes through G. Hand-holes E are very essential, as the integrity and life of the jacket depend largely upon the care that is given to keep its interior free from mud and lime-scale. The hearth M rests upon the drop-bottom P, and is built up of fire-brick and clay. The slag-notch is at L, and the tap-hole for the metal at O. The entire furnace rests on the four short columns R, and is covered by the hood H leading to the stack K. A furnace of this description, 42 to 46 inches in diameter at the tuyeres and 6 to 9 feet high from tuyeres to charge door, smelts from 40 to 80 tons of ore per day. It is usually driven by a No. 4½ Baker blower, running 100 to 120 revolutions, and furnishing some 2,000 cubic feet of blast per minute.

In rectangular wrought-iron jackets, the shaft may be divided into narrow sections, as with cast-iron jackets, or each side and end of the shaft may be formed by a single jacket. Figs. 33 and 34 show this latter form of construction, though in this case there are two tiers of jackets, one above the other. The rectangle is 32 inches by 72 inches at the tuyeres inside. The upper jackets B are supported by the columns L, while the lower jackets A rest on

* Figs. 29 to 34, with a portion of the accompanying descriptions, are taken from the valuable paper of A. F. Wendt, in Vol. XV. of the *Transactions American Institute Mining Engineers*.

Fig. 29.

Fig. 30.

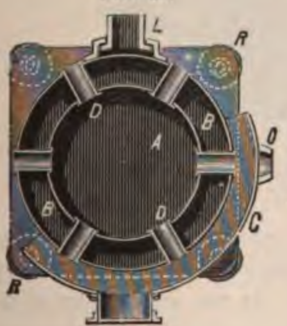


Fig 31.

Fig. 32.

the bed-plate carried by the posts K. HH are the tapholes and M the slag notches. The upper part of the furnace is surrounded by the shell O, and contains a charging-bell and hopper which is worked by the levers V. This has been replaced by a simple

FIG. 33.

hopper. The capacity of such a furnace varies so completely with the ores and blast used, that it is impossible to speak of it accurately except for known conditions. It can smelt from 60 to 110 tons ore per 24 hours.

Two main objections are occasionally found to these large rectangular wrought-iron, or soft steel water-jacketed cupolas. These are the corrosion of the upper portions of the jacket by material containing sulphate of copper, and the buckling or distortion of

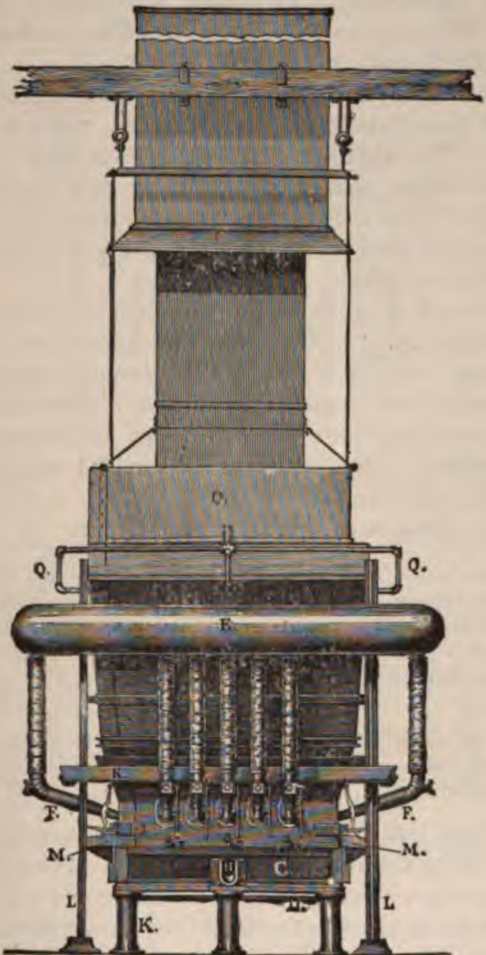


FIG. 34.

the inner shell of the large jackets that form the long sides of the rectangle. Herreshoff has long substituted copper for iron for the inner shell, to obviate the first difficulty, while the second is overcome by dividing up the long sides of the furnace into several sections. This has nowhere been more perfectly done than in a late furnace erected at W. A. Clarke's Verde mine, near Prescott, Arizona. The original idea of the furnace was given by Mr. J. L. Giroux, the details being worked out by Fraser & Chalmers, whose long experience in such work has enabled them to steer clear of the difficulties so often encountered in new designs. Plate IX. shows this furnace in detail.

It consists of three tiers of sectional water-jackets, extending from the cast-iron base-plate to the charging door, which is 9 feet above the tuyeres. The middle tier of jackets has a bosh near the bottom, and the upper tier is so set that it tumbles in toward the tunnel head. Thus the side-walls are contracted at the hearth and at the tunnel head, and widen out at the middle of the shaft. The end walls are vertical.

The inner shell of all the jackets is made of $\frac{3}{8}$-inch copper, and the outside shell, of $\frac{5}{16}$-inch flanged steel, the two shells being stiffened by stay bolts that pass across, through the water-space. These stay bolts have caused no leakages. There is a partitioned wind-box containing 10 tuyere openings, the castings being of phosphor bronze. Each opening is provided with a deflecting nozzle, and a ball-valve to control the blast.

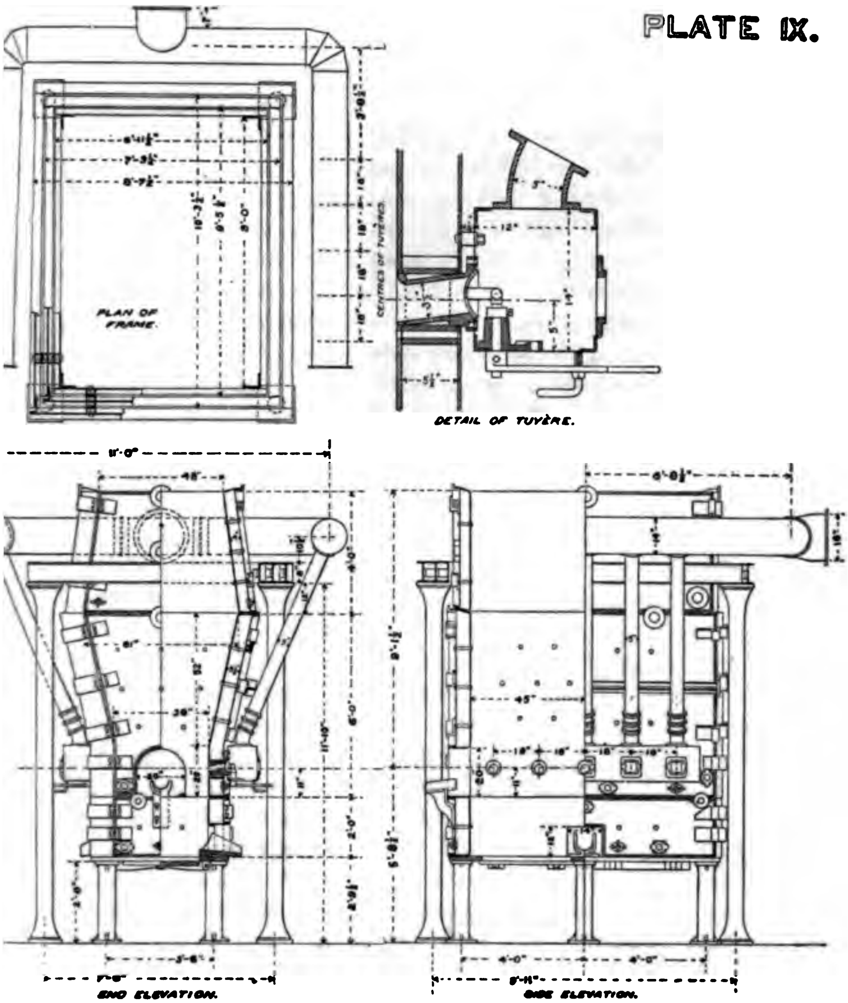
The inside dimensions of this furnace are:

Length	90	inches.
Width at tuyeres	36	"
Width 5 feet above tuyeres	61	"
Width at tunnel head	48	"
Width of water space in upper tier of jackets	4	"
" " " in two lower tiers	$5\frac{1}{2}$	"
Total weight of jackets, base-plates, I-beams for supporting brick-work, etc	24,000	pounds.
Total copper in furnace	10,181	"

Unless a metallurgist has had long experience with the various forms of water-jacket and the various details belonging to them, and unless he knows exactly what he is doing and why he is doing it, it is far safer to trust to the established manufacturers of this apparatus, than to attempt to originate any improvements that diverge very radically from the regular type. Inventing is one of the most expensive amusements belonging to metallurgy, and should be generally left to those individuals who are led toward it by experience and talent.

Having obtained and erected the furnaces and blowing plant,

PLATE IX.



COPPER-LINED WATER JACKET
FURNACE.

there remains only to make the water and blast-connections, put in and dry the furnace bottom, and prepare and heat the forehearth. Care should be taken in, planning the furnace, that there are no narrow passages or pockets in the water-space, where sediment and scale can collect. Otherwise, these will quickly block up and burn out. Such places may be between the tuyere-castings and the lower ring of the jacket, or between the slag notch and lower ring. If any such exist, it is an excellent plan to drill a small hole from below into the narrow spot and put in a one-half, or three-quarter-inch pipe, through which a constant current of feed water forces itself upward under pressure, and thus prevents the collection of sediment or scale. This does not supersede the frequent opening of the hand-holes and inspection of the water-space for sediment or scale, and a frequent blowing-off of the jacket under all the pressure possible, or assisted by the introduction of live steam.

Water-jacket furnaces have two classes of bottoms. In one class, the bottom is made as in Fig. 30, being a thick mass of brick and clay, and dependent for its integrity on the comparatively high grade, of the product made. As every metallurgist knows, rich matte, or metallic copper, tends to fill up a hearth rather than cut it out, and in a furnace producing such material, there is no need of having the water-cooling extend down to the bottom of the hearth, or even to provide a water-cooled slag-notch, save under exceptional circumstances. Where the product is a matte of lower grade and especially if made in considerable quantity, such a bottom as the one shown in Fig. 30 would soon be cut through and the hearth destroyed. By extending the water-jacket down to the bottom plate, the sides of the hearth are rendered safe, but the bottom is still vulnerable, and, in fact, in running at high speed on a matte of 35 per cent. copper or less, the bottom plate would be eaten through and the contents of the hearth would escape within an hour or two. This catastrophe only takes place from the cutting-down action of the slag and matte at the slag-notch; and if this notch be jacketed all around so that its level cannot be changed, as in the Herreshoff furnace, Fig. 27, the hearth of the furnace up to the lower edge of the slag and matte-notch G, will always contain a pool of stagnant metal that is scarcely affected by the hot products resting upon, and flowing over, its surface. The bottom of this quiet pool of matte is in contact with the thin layer of fire-brick which alone separates it from the cast-iron bottom-plate E, and thus radiates heat so rapidly that it soon becomes chilled and practically forms the bottom proper of the furnace. If the furnace is run hotter or faster, or with a greater proportion of matte, or on a matte of lower grade, an inch or two of the surface of this artificial bottom will be cut away, and this will continue until the radiation of heat through the thinned bottom exactly balances the accession of heat from the smelting, when it will again become stationary. This building-up and cutting-down of the bottom is entirely automatic and requires no attention or assistance from the metallurgist.*

The drying of the thick bottom, as in Fig. 30, may require 18 hours or more, as it is undesirable to leave any moisture to form steam. Where this occurs, a boiling action of the molten products is set up that is apt to result in loosening, or partially destroying the bottom.

The thin bottom, as shown in Fig. 27, is usually dried for a few hours, a small wood or coke fire being maintained in it, and the ashes removed from time to time that they may not form a non-conducting layer between the heat and the bottom. But even this slight drying is hardly essential, as the object of the single layer of brick that forms the bottom is merely to keep the hot metal away from the bottom plate until an artificial bottom is built up by the chilled matte and slag.

In smelting oxidized ores for black copper, the bottom is made deep enough to form a small crucible for the accumulation of the metallic product, which has too high a melting point to attempt to collect in an unheated outside forehearth. With this exception, however, it is the ordinary practice in the United States to allow no accumulation of molten material inside the furnace, but to effect the separation of matte and slag in an independent outside forehearth, or well. Next to the introduction of the water-jacket furnace, I regard this practice of universally settling the matte outside of the furnace, and thus removing all material as soon as possible from the hearth, as the most important advance of this generation in the blast furnace treatment of copper.

It may be assumed that the ordinary difficulties experienced in running a furnace with brick hearth built up from the ground and with interior crucible or sump, are mainly due to its filling up

* This principle of automatic regulation by radiation has a wide practical bearing in metallurgical operations. It is also a good example of the advantage of accomplishing an object by enlisting natural forces in our behalf instead of struggling to oppose them.

'ith sticky, half-fused products that become more and more diffi-ult of removal, and finally accumulate until the furnace must be lown out.

I need hardly consider the opposite condition of affairs where ie hearth is cut away and deepened, until, in some of our large rick furnaces it may contain 25 tons, or more, of matte. This ccurs only when producing large quantities of very low-grade iatte, 8 per cent. to 15 per cent. copper, and usually happens uring the reducing smelting of raw pyrites fines. If the hearth nd foundations of the furnace are properly constructed, it is best) let matters take their course, feeling sure that when the matte as cut its way down deep enough to make the radiation below qual to the accessions of heat from above, it will cease burrowing f its own accord.* This leaves a permanent bottom, containing erhaps 40 tons of 15 per cent. matte, or 12,000 pounds copper, 'orth perhaps 7 cents per pound in this condition, or $840 in all. .t 6 per cent. per annum, this amounts to 14 cents per day, or bout one-seventh of a cent per ton of ore smelted, which is not n extravagant price to pay for the luxury of a bottom that re-uires neither renewals nor repairs.

Hence, we may fix our attention on the *filling-up* rather than ie *cutting-down* of the crucible. While the accretions that so 'equently form in the hearth of a furnace with interior crucible re often termed sows, salamanders, or bears, it is seldom that iey are entited to these designations, which are more correctly

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* The principle of automatic regulation by radiation is again illustrated in iis practice. With competent and experienced furnace-men there is scarcely limit to the time which such a bottom will last, being constantly torn down id built up by its own internal processes. It is the furnace-men's duty to ssist these matters by various well-known means at their disposal, among 'hich the commonest are:

Using an excess of pyrites and a heavy blast, so as to make a poorer matte nd cut down a bottom that has grown too high.

Using less pressure of blast, but larger tuyeres, to effect a more forcible xidation of the pyrites, and thus make a thin, ferruginous slag, and a richer nd scantier matte, which will soon build up a vanished bottom. Or, if the ottom seems to be cutting down beyond all bounds, allowing the furnace to tand without blast for several hours, during which time radiation from the rucible will be going on without any accession of heat from above. This is a ery certain means, and will soon lay the foundations of a solid hearth, which , built up still more by the richer and more infusible matte produced when ie blast is again let on to the charge which has been slowly roasting during ie period of repose.

applied to accretions consisting mainly of metallic iron.* These are generally of gradual growth and are produced most freely in furnaces where the smelting is slow in comparison with the hearth area where there is a high ore column, a contracted hearth (boshes), and *a scarcity of iron or other bases.* Paradoxical as it may seem at first glance, a scarcity of iron (or proper bases) in the slag causes iron in metallic form to be separated out from this same slag. Yet the reason is quite obvious. By withdrawing iron from the slag, we decrease its fusibility and raise its smelting point. Now a siliceous slag with high melting point, produced in a furnace intended for a more fusible mixture, brings about slow smelting, a rising of the heat toward the tunnel head, a powerful reducing action, and, in a word, inaugurates on a small scale the condition of affairs prevailing in furnaces devoted to the production of pig iron from ores. We have not a sufficiently high temperature nor reducing power to form the ferrous carbide that we know as *cast iron*, but we can produce an infusible wrought iron with the greatest facility.

Such conditions are rare in America, as rapid driving and the production of fusible slags by the avoidance, or mechanical concentration of siliceous or aluminous ores are opposed to the formation of metallic sows. Hence, the accretions that we find in our furnaces are apt to be mixtures of magnetic oxide of iron with infusible slags, indefinite compounds of baryta, zinc oxide, etc., with which is usually interspersed a quantity of very basic matte; that is to say, a matte with too large a proportion of iron and too little sulphur. These accretions are very hard to break up, even when outside the furnace, and are so slippery and intangible when at a high temperature, that it is very difficult to drag them out of the interior. We escape this filling up of the furnace, and the serious labor and delays attendant thereon, by transferring the settling process to an outside crucible (forehearth or well), which is not only accessible,

* A sample of borings from such a chill, analyzed for the writer by Mr. A. F. Glover, Ph.D., had the following composition:

Sulphur	4.64	Slag	0.78
Copper	9.80	Nickel and cobalt	0.81
Iron	82.70		100.26
Carbon	1.12		
Arsenic	0.41		

and thus easy to clean out and repair, but which can be removed and replaced inside of an hour.*

The four main causes of trouble and delay in the running of copper blast-furnaces, and the means generally adopted in the United States for the avoidance of these inconveniences, are:

TROUBLE.	REMEDY.
1. Destruction of lining.	Water-cooled walls.
2. Choking-up of shaft by accretions.	Metallic water-cooled shaft, which permits their easy removal.
3. Burning-out of crucible and bottom.	Self-created bottom, automatically regulated by radiation.
4. Filing up of crucible or hearth by sows, or other accretions.	Permitting all molten material to run out of hearth as soon as it can, and thus transferring possible accretions to an external and exchangeable forehearth.

The blowing-in of a modern water-jacket copper furnace, on known ores, whether small or large, would be scarcely worth alluding to, were it not that traces of the anxiety and importance that once attached to this operation still hang about it.

In starting a *brick* furnace there is a large mass of material to be warmed up, and, above all, most of this material can be destroyed by an excess of heat, or by a very trifling want of skill in fluxing or management. In a water-jacket, however, the only extra caloric required for warming up is the few heat units necessary to raise the jacket-water to its normal temperature, and to prepare the cold bottom for the molten matte and slag that are soon to cover it. But the feed-water and bottom have probably both been heated up by a preliminary drying fire, and a few inches of hot, low-grade matte will do more to get the bottom in proper condition than hundreds of pounds of coke. Hence, in blowing-in we have no use for any extra fuel except to heat the first few thousand pounds of slag and matte sufficiently beyond their proper normal temperature to provide enough heat to warm the bottom, and especially the external forehearth, up to its regular condition. As the forehearth has already been brought up to a red heat by means of a bushel or two of coke or charcoal (aided, perhaps, by a light blast through the tap-hole at the side), the amount of heat to be abstracted from the molten products to bridge the space between red-heat and the normal white-hot condition of the forehearth

*The subject of forehearths is sufficiently important to demand a separate section for its consideration.

is very small. Beyond the fuel necessary to supply this slight amount of heat, every pound of extra coke is a positive and serious detriment in various directions. The two most obvious evils are: The waste of money in consuming coke to uselessly heat the jacket-water, and the much more serious matter of reducing iron out of the slag by the high temperature and powerful reducing action arising from the excess of fuel.

This over use of coke in blowing-in a water-jacket (where one loses the wholesome restraint imposed by the fear of damaging the lining), is so common and serious an error that it seems worth while to illustrate it by an example.

Some years ago I was present at the starting of a large water-jacket furnace in Southern Arizona. The ores were pure carbonates and oxides, and the slag was to be rather siliceous and low in iron; lime and alumina being more accessible than were ferruginous ores. The slag had been carefully calculated, and appeared to be a feasible one, though the former owners of the mines had run a highly ferruginous slag, exhausting all the cupriferous hematite that could be found in the neighborhood. The furnace had been in blast six hours when I first saw it, and presented a very sickly appearance. The slag was only red-hot and very scanty, and, apparently, extremely siliceous. No copper could be found on trying the tap-hole, and all 10 tuyeres had long noses that united at the center of the shaft, and through which not the feeblest glow of heat could be seen. The charge sank extremely slowly and irregularly, the jacket-water was almost boiling in spite of a full supply through an ample feed-pipe, and the heat was mounting to the tunnel head. There were obviously strong prudential reasons against blowing out and starting afresh.

As is usual in such cases, the furnace had been started with an enormous excess of coke, and even after six hours running, only one-half the charge that this coke was expected to support had been reached. Yet the furnace-men were clamoring for a few "empty charges" (coke without ore) in order to heat up the slag and melt out the solid mass that pretty nearly filled the hearth and lower portion of the shaft. If the furnace had been a small one it could scarcely have been saved, but it requires a considerable amount of time, as well as metallurgical skill, to completely freeze up one of the long, rectangular shafts now in such common use, and there was still hope.

A totally new departure was agreed upon. Every alternate tuyere was plugged, the blast was reduced to a quarter of an inch of mercury, the coke charge was maintained at 300 pounds, but instead of small charges of ore they began on 3,000-pound charges of the old (rich) ferruginous slags. These were continued for five charges, when one-half of the slag was replaced by ore, and later, ore was gradually substituted for the remaining 1,500 pounds of slag at the rate of 40 pounds ore for 100 pounds slag, so that the normal charge became 2,100 pounds ore to 300 pounds coke, with the addition of the 2 or 3 per cent. of foul slag made by the furnace. For some two hours after the change, things looked very bad. The slag stopped running almost completely, and the wind blew through the cold noses of the tuyeres as though its only effect were to remove what little warmth still remained in the siliceous skeleton inside. But finally the plugged tuyeres began to brighten one by one, and then it was evidently only a question of time. It is almost as hard to damage an improving furnace as it is to better a sickly one. The bright tuyeres were put in blast as they became fit for it, and their chilled neighbors were plugged in turn, until eventually there were no noses left, and as the heavy charges of basic slag came down and swept the siliceous chill before it, the furnace went to the other extreme, and it was impossible to handle the slag with the 12 pots provided. Some holes were dug to one side in the floor, and many tons of slag run into them, to be later hoisted out with a crane. A large bed of black copper was obtained before it was possible for any of the ore that had been charged after the slag-charges, to get down. The now thoroughly hot furnace required only 1,200 gallons of jacket-water per hour, whereas in its frozen condition it was using something more than three times that amount.

A slight alteration of the ore-mixture was found advisable, and eventually the furnace settled down on to a charge of 300 pounds coke, 2,250 pounds ore, 100 pounds old slag, and such foul slag as was daily made in the process. *The first tap of black copper contained 32 per cent. iron.* After regular work had become established the iron fell to about 4 per cent.

In the light of the preceding pages it is not difficult to see what was occurring at the start. There were several errors in judgment. In the first place, the furnace was started on ore instead of on a ferruginous slag-charge. This is not absolutely necessary, but it makes things much more comfortable to start with a charge or two of good, basic slag, and when blowing-in on new, untried ores, it is doubly important to do so. Again, the normal, calculated charge was used from the outset, whereas it is always wise to start a siliceous charge so that the slag shall contain some 5 per cent. less silica than it is eventually intended to keep it at. It is easy enough to make a basic charge more siliceous, but very tedious and difficult to render a chilled, slow-running, siliceous charge more basic.

The third and greatest mistake was the use of too much coke. The charge only required some 12 or 13 per cent. of coke to melt it, and the extra 12 per cent of the 25 per cent. actually used could only expend itself in heating unnecessary jacket-water and in reducing iron out of the slag. This was exactly what occurred. The ample feed pipes could barely supply the jackets with sufficient water; the surplus heat and strong blast forced the combustion to gradually ascend the shaft until, on my arrival, the coke was burning fiercely at the charging door. The metallic and most fusible portions of the ore were liquated out in this intense heat; some of the iron was reduced to the metallic form, and by the time the slow-sinking column had reached the proper smelting zone, it was merely a dry and highly siliceous skeleton, with all the coke burned away in the upper regions of the furnace, and ready to chill into a solid mass as soon as touched by the blast. If the furnace could have been turned upside down, there might have been some chance for it, as nearly all the heat was in the upper part of the shaft and the hearth and tuyere zone were black and cold. The metallic copper liquated from the ore above trickled down, alloying itself with the reduced iron, and finally set as a solid mass in the chilled hearth. The only slag obtained came from the slow liquation process that was going on several feet above the tuyeres, and trickled down just back of the breast, where it was protected from the blast.

Evidently, two things had to be done promptly if the furnace was to be saved. These were:

1. To get the heat down from above, where it was doing mischief, into the tuyere zone, where it belonged.

2. To dissolve and remove the partly infusible skeleton of silica that blocked up the tuyeres and smelting zone.

These objects were accomplished:

1. By greatly reducing the blast pressure to prevent unnecessary chilling of the half-fused, cokeless masses in the hearth, and the driving of the heat any higher up the shaft. At the same time,

every alternate tuyere was plugged, to give its chilled nose a chance to melt away.

2. By allowing the ore-column in the shaft to sink several feet, and then adding all at once several full charges of coke and ferruginous slag.

This cooled the heated shaft at once (the upper portion of it being of fire-brick), and permanently suppressed all fire on top. This suppression resulted partly from the light blast, partly from the now cooled walls which could not ignite the coke in the upper zones, and most of all from supplying the new coke with all the work that it could do, so that it had no heat left with which to do mischief. The basic slag that was charged could not be robbed of its iron, as it smelted into a thin liquid long before it could be subjected to any dangerous reducing action, and, trickling down toward the hearth in a multitude of thin streams, permeated the quartzose skeleton opposite the tuyeres in every direction. Taking up additional silica with avidity, it acted very much as a stream of hot water would act upon an already fissured and rotten mass of ice. The noses of the plugged tuyeres, not having their heat any longer absorbed by the blast, were the first to soften and disappear, and when they became reasonably free, and the fresh coke had a chance to get down in front of them, they were put in action and their neighbors were plugged and given the same chance to recuperate.

As soon as the fresh coke and the ferruginous slag got down to the tuyere level the action became very rapid, and the great chilled mass below, added to the basic flux from above, made a most copious slag-flow that thoroughly warmed the hearth and forehearth and heated up the frozen block of copper that still encumbered the crucible, and that was strongly alloyed with the iron that had been stolen from the mixture during the first period of the blast. This copper, owing to its great heat-conducting capacity, remained solid until its entire mass had been brought to the point of fusion, when it melted all at once and yielded the large bed of ferruginous copper already mentioned.

Nothing now remained but to regulate the mixture so as to produce the cleanest and most advantageous slag possible under the conditions, and then to gradually increase the ore-charge until the coke was carrying every pound of ore that it could possibly smelt. Then, and not till then, was the furnace doing its work properly and employing its fuel in smelting ore instead of in heating water and reducing iron.

If the above description of a most common, and often disastrous, state of affairs seems a little too minute for a text-book of this description, it must be recollected that the illustration just given deals with the major portion of the difficulties encountered in running a water-jacket furnace.

The amount of water required by a water-jacket furnace, cooled from hearth to throat, depends so much upon the local conditions, that it is impossible to lay down any fixed rules for its consumption. It will depend mainly upon the following factors, arranged according to their influence:

Whether the ores smelted are of a nature to form a uniform protective coating upon the walls of the shaft and one that does not grow so rapidly as to require frequent removal by barring.

Whether the coke used is kept supplied with all the ore it can possibly take care of, so that it may have no energy left to waste in heating jacket-water.

Whether the protective coating consists of substances that are good conductors of heat, or the reverse.

Whether the ore is granular, or contains too large a proportion of fines, in which latter case it will be necessary to frequently allow the ore column to sink deep in the shaft, and thus expose the jackets to powerful, though temporary, heat.

Upon the pressure of the blast and the skill exercised in the general management of the furnace, so that the heat shall not keep constantly mounting toward the tunnel-head.

Upon the specific heat of the slag.

Under ordinary conditions and proper management, the maximum amount of feed-water required is shown in the following table, compiled from personal experience, and referring to furnaces when run up to full capacity.

Hearth Area. Square feet.	Water per hour while blowing in and out. Galls.	Water per hour during normal running. Galls.
3	900	460
5	1,200	600
7	1,450	950
9.5	2,200	1,100
12.5	3,000	1,300
18	4,000	1,500
24	5,000	1,800
30	6,000	2,000
36	7,000	2,200

These figures refer to a supply of fresh water; but where the same water is used over and over again, about 25 per cent. more is required to make up the loss by evaporation, etc., in a 36-inch furnace in the dry, hot climate of Arizona.

FOREHEARTHS.

The most important, and apparently least understood, portion of a copper-matte blast-furnace is the forehearth.

I have already referred to the great advantage that is gained by allowing the molten products to escape from the hearth as soon as formed, thus transferring the settling operation from the inside to the outside of the furnace. The burning-out of the crucible, the formation of sows and accretions, and the various difficulties that are inseparable from the use of an inside crucible are thus avoided, and even if they are only transferred from the interior to the exterior of the furnace, it is an enormous advantage to have them where they are distinctly visible and can be got at and remedied at once.* Before the days of external forehearths, more than 75 per cent. of the delays and difficulties encountered in running a blast-furnace were connected with troubles and uncertainties regarding the condition of the hearth and crucible, and a furnace was often run at a loss for a considerable period, in the hope that it would "come round all right eventually" and save the cost and delay of blowing out and putting in, and drying, a new crucible.

In the modern practice these troubles are transferred to the forehearth, and with proper arrangements it takes only 30 to 60 minutes to replace it with a fresh one.

There are two objections sometimes urged against the abolition of an internal crucible or sump, though I have never heard either of them cited by men who had had a varied experience in the use of suitable external settlers. It is sometimes alleged

1. That there is a loss of heat experienced by giving up the internal crucible.

2. That the matte is more perfectly settled inside the furnace.

While it may be possible to select isolated cases in which either

* It must be remembered that I am referring entirely to American conditions, where 100 tons or more of ore are smelted in the furnace daily, practically without slag, and if at all possible, without flux, and that but two products are allowable: A matte of good grade and often containing considerable amounts of silver and gold, and a slag that must be poor enough to go over the dump.

or both of these objections might be valid, my own experience contradicts them completely under ordinary conditions. Where a siliceous slag is made from a hard-smelting mixture and but a small quantity of high-grade matte is produced, it is often a question whether the inside crucible might not be more economical. The smelting at Mansfeld in Prussia is a typical instance of this kind. The ore, after being burned in large heaps to remove the bitumen of the shale, is smelted in large, high blast-furnaces, with hot blast and under conditions much resembling those present in the production of pig iron from its ores. About 17 tons of the ore are concentrated into one ton of matte, having about the following composition:

Copper	45	per cent.
Sulphur	24	"
Iron	20	"
Zinc	4.5	"
Lead	1	"
	94.5	"

with small amounts of manganese, cobalt, nickel, and silver.

An ordinary slag from the Saengerhausen smelter had the following composition, according to Heine:

Silica	53.83	per cent.
Alumina	4.43	"
Lime	33.10	"
Magnesia	1.67	"
Ferrous oxide	4.35	"
Cuprous oxide	0.25	"
Fluorine	2.09	"
	99.74	"

With this poor ore, high rate of concentration, comparatively rich matte, and extraordinarily siliceous slag (probably the most siliceous slag made regularly anywhere in the world, in blast-furnaces), an interior crucible is, no doubt, essential. Even here, however, it causes more or less annoyance and delays, as well as the production of nickeliferous sows, which are sold at the valuation of pig iron.

But for anything approaching ordinary conditions, the two objections cited are not valid so far as I am competent to judge.

The first objection, that the use of a forehearth causes a loss of useful heat, is not difficult to meet, as it can be almost anywhere decided by actual trial at very slight expense. I have tried the experiment on several occasions and with considerable care, and have never been able to effect any saving in fuel by retaining the matte in the furnace in a crucible, though I have very frequently witnessed a decided increase in its consumption from irregularities brought about by this practice.

I am not aware that any one claims any demonstrable saving in fuel in the actual smelting operation from the use of the interior crucible. The ordinary statement is, that by retaining a large body of matte in the furnace, the bottom and hearth are kept hot and in good condition.

To this I reply, that all I demand of a bottom is to have it furnish me a solid and slightly inclined surface on which my molten products can run out of the furnace as fast as they are formed, and that it is a matter of entire indifference to me if its temperature is 20 degrees below zero, as I feel entirely confident that the matte and slag will simply chill enough to form a non-conducting crust sufficiently thick to prevent the cold bottom from stealing heat from the constant and powerful stream of molten products.

A *damp* bottom is a decided evil, as the escaping steam bubbles through the slag and, aside from its cooling influence, produces serious trouble mechanically. But a *cold* bottom need never be feared in a fast-running furnace with independent forehearth.

As regards the second objection, which has to do with the settling of the matte, I will frankly admit that many forehearths are so constructed and managed as to lose more copper than would be the case with an interior crucible, always providing that we could guarantee the latter against irregularities. But this loss in matte comes almost entirely either from a badly-arranged forehearth, or from want of care and skill in managing it. If a furnace with interior crucible were run with the same carelessness and nonchalance that is so often displayed in running one with forehearth, the question would soon arise, not how to save the matte, but how to save the furnace. Because a system is so easy and comfortable as to frequently lead to carelessness and abuses is no valid reason for discarding it and refusing to profit by its advantages. The small proportion of copper that exists in the slag as cuprous oxide will not be saved (by any ordinary means) either in a crucible or in a forehearth, and the matte globules themselves can be settled as perfectly in the one as in the other.

The forehearth in its simplest form consists simply of some sort of vessel or box external to the shaft of the furnace, into which the molten products can flow, and separate according to their specific gravity. A simple settling-pot is a forehearth, though a rude and unsatisfactory one.

This subject is so important to the metallurgist that I shall describe the main forms of forehearths in detail, premising that their variety is considerable, and that each form described stands merely for a general type that has many variations. I shall neglect the rudest and simplest forms of forehearth, as they are inefficient and can also be easily deduced from the more perfect ones.

Perhaps the simplest form of efficient forehearth is a rough rectangular box made of four cast-iron plates set on edge on a cast-iron base-plate, the latter being mounted on wheels, that the whole structure may be easily removed or replaced. The fastenings of the plates should be as simple as possible, being usually confined to a couple of rods that connect the extremities of the two longer plates, the short end-plates being retained in place by a vertical ledge cast on the side-plates.

There is a removable cast-iron slag-spout at one end and a vertical slot for a tap-hole at the side. When the matte is low-grade and plentiful, and thus likely to burn away the lining of the forehearth, the tap-hole should be situated near the end of the box farthest from the furnace. When the matte is scanty or tolerably high-grade, and thus liable to chill, the tap-hole should be placed nearer the furnace, that it may receive all the heat possible from the molten stream that is constantly entering the forehearth.

The lining of this, or any, forehearth must be suited to the local conditions. Where the slag is siliceous or the furnace is small or runs slowly, or the matte scanty and high grade—45 per cent. or over—the material for the lining may be found in the nearest bank of clay or loam. This, mixed with chopped straw or horse manure, as hair is mixed with mortar for plastering walls, forms a cheap and satisfactory lining. Its sole duty is to keep the molten contents from cracking or burning the cast-iron forehearth plates until they begin to form a protective crust upon the bottom and sides, that will gradually continue thickening until the cavity of the forehearth becomes too small to act as a suitable settler, or it becomes too difficult to drive a bar through the tap-hole. This may occur in a few days, weeks, or months. Perhaps the average life of a good sized forehearth under favorable conditions may be put at three weeks.

When the matte is of lower grade and abundant, and the slag ferruginous, such a lining would be quickly destroyed, and it becomes necessary to use 4½ inches of fire-brick, laying the brick as carefully and with as thin joints as would be done in the walls of a furnace. Even this brick lining is occasionally insufficient, and where a very large proportion of matte is formed, and especially if the furnace is run very hot in order to store up a large charge of matte for the Bessemer converters, and at the same time keep this great body of matte at the high temperature required for tapping into a ladle and conveying and pouring it into the converter, it is found necessary to greatly increase the size of the forehearth. This is done in the Boston & Montana smelter at Great Falls, Montana, where charges of 5 tons of molten matte are required for the converters.

The charge smelted in these blast-furnaces is exceedingly hot and fusible, consisting of 50 per cent. of a mixture of first-class ore and concentrates assaying over 20 per cent. copper, 36 per cent. converter slag, 14 per cent. limestone, with the addition of a certain amount of refinery slag. The furnaces are 36 inches by 108 inches at the tuyeres, and smelt about 110 tons of the mixture daily. The concentration of the ore is only a little more than 2 into 1, the added slags enriching the matte considerably, and over 25 tons of 50 per cent. matte is produced daily at each furnace.

To enable the forehearth to store up some 5 to 8 tons of this matte, it must necessarily be large; and to prevent so much fiery material from cutting out and breaking through the lining, the forehearth has been made shallow and has gradually been increased in diameter (its form being circular), until enough surface was gained to chill the matte at the circumference just sufficiently to prevent its bursting through the lining. This is another example of the great principle of automatic regulation by radiation, that has already been prominently noticed. At first glance it might be supposed that the larger the forehearth the smaller would be its radiating surface in proportion to its capacity; and this supposition, regarded simply as a mathematical proposition, would be quite correct. But other factors modify the conditions. The cutting-out of a forehearth is effected largely by the direct influence of the constant white-hot stream of metal from the furnace; and the farther the walls of the forehearth are removed from this center of heat, the less will be its influence upon them. Besides, in these large forehearths, the depth of the metal is very slight compared with its extent, and the effective radiating surface is thus very largely increased.

These forehearths are now made 10 feet in diameter over all, the former 8-foot ones having burst out too frequently. They consist of a cylinder of boiler iron 10 feet in diameter and without top or bottom. The bottom is formed by a course of 4½ inches of fire-brick laid on a foundation of rammed clay. The lining consists of two 4½-inch rings of fire-brick, so that the forehearth, when completed, has a clear diameter of nearly 8½ feet. There are two tap-hole slots; one, halfway up the side; the other near the bottom. Of course there is a tap-hole slot in the boiler-iron casing and a corresponding notch in the brick lining, but the tap-hole proper is formed and kept in condition by a plate of copper as high as the forehearth, 2½ inches thick at the top, and tapering to 1½ inches at the bottom. This slab of copper is simply slipped between the iron casing and the brick lining and is pierced by a 1½-inch circular hole that corresponds to the tap-hole slot. Radiation again comes into play, and this uncooled copper slab answers its purpose satisfactorily and keeps a free and easily controlled tap-hole. As a chill is very apt to form at the interior orifice of the tap-hole, especially as the forehearth gradually fills up with accretions, a steel bar is kept constantly in the taphole. By driving it slightly with a hammer from time to time, its point is kept about even with the inner surface of the slowly increasing chill, so that the latter is easily penetrated by a few solid blows when the time for tapping has arrived.

The short spout that conveys the molten products from furnace to forehearth is of water-jacketed boiler iron, as unprotected metal of any description would be destroyed within an hour by the powerful stream of matte and slag. The forehearth lasts about a month, and handles some 3,500 tons of melted material before requiring to be replaced.

It is important with this, as with every roofless forehearth, to get it well protected at the start with a proper covering that shall retain the heat and guard its contents against too rapid chilling. This is best effected as follows: A hot fire is kept in the forehearth during the blowing-in, and the slag-notch of the furnace is not opened at all until the interior is filled pretty nearly to the tuyeres with molten products. The furnace should always be started on a somewhat basic and easily-fusible mixture, and one that will produce a pretty large proportion of matte. Just before the notch is opened, the forehearth should be scraped tolerably clean of ashes and cinders, and after the first flush of slag has run out of the furnace, the surface of the molten mass in the forehearth should be liberally covered with light wood, the hot flame from which will prevent a stiff slag-crust from forming on the gradually rising bath. When the forehearth is full, it is well to dam its slag-spout with a lump of clay and allow the bath to rise even above its surrounding walls, breaking up the slag-crust all over the top with a bar, to permit its free elevation. At length when the forehearth is brimming full and the center has even risen two or three inches above the side walls, the crusted surface of the bath is evenly and thickly covered with a non-conducting layer of coke dust. This crusted roof of slag is of the greatest importance to the integrity of the hearth, and should not fall in when the matte is removed by tapping. If it should grow too thick or too thin, we only have to apply our familiar principle of regulation by radiation and add to, or take away from, its protecting layer of coke dust.

It is the constant attention to just such apparently trifling details as these that enables some foremen to run a furnace without delays or difficulties, while others have frequent stoppages and constant trouble and hard work for their men.

The next type of forehearth is one rendered familiar to many smelters by Herreshoff's patent water-jacket furnace. It is a much more ingenious, and, for certain conditions, a much more perfect device than any of those hitherto described. Run by experienced furnace-men and on proper ores, it enables a furnace to run as nearly absolutely without stoppages or without producing foul slag, as is well possible.

To understand Herreshoff's furnace and forehearth, it will be necessary to turn to Figs. 27 and 28 on page 265.

The furnace here shown is rectangular in shape, with corners rounded, and the lines between the corners slightly curved or of convex shape. The height is 10 feet, width 3 feet 7 inches at the bottom, and 4 feet 7 inches at the top, by 6 feet 4 inches length at the bottom, and 7 feet 4 inches at the top. The water-jacket is exceptionally narrow, having a water-space of only 2 inches.

Referring to the cuts, A is the body of the furnace; B a ring 2 by 2 inches, to which the plates of the water-jacket are riveted. At the top C, the outer plate is flanged 2 inches, and the inner plate 4 inches, and the flanges then riveted. The bottom of the furnace E is a dished cast-iron plate 1½ inches thick, fastened to the ring B by tap-bolts. This permits the dropping of the bottom if required. The legs F are bolted to the ring B on the outside of the furnace, thus not interfering with the dropping of the bottom. The hole G is the outlet of the furnace for both slag and matte. It is 9 inches high and 7 inches wide and made by riveting the wrought-iron frame H into the shell of the furnace. The furnace is blown by 13 tuyeres, five on each side and three on the back. They are placed 26 inches above the bottom plate, and are 2 inches in diameter.

The construction of the furnace proper is practically identical with that of a former round furnace, but the forehearth is considerably changed. In the round furnace the forehearth was floored with a layer of slag-wool and brick as described. A brick lining was also used. The bottom of the brick lining was some 12 inches below the outlet from the jacket. Experience proved that this bottom invariably chilled to a level with the bottom of the opening to the furnace. The cutting of the brick lining at a higher level also gave occasional trouble. Both these faults are avoided in the present construction. The former, by raising the forehearth on high wheels N, and making the floor of the bottom lining within 2 inches of a level with the bottom of the inlet L The latter, by entirely casting aside fire-brick lining and depending on the circular cast-iron water-jacket K. The tap-hole R in the shaft of the furnace is used only when blowing out to tap the furnace clean, or sometimes, for such small quantities of black copper as may be accidentally made. In the forehearth, the tap-hole O is the one commonly in use. It is made of copper, bolted to the iron body of the forehearth and is water-jacketed similarly to the "Lührmann" slag tuyere of iron furnaces. The manner of operating it is also similar. M is the slag-spout; W, a brick-lined, dish-shaped movable iron cover of the forehearth. When smelting, the well or forehearth is wheeled up against the furnace, as shown in the cut, and a very small amount of wet fire clay is placed on the iron faces surrounding the holes G and L, in order to make a tight joint between them.

In practical operation, after the furnace has been properly charged, the blast is let on. The first cinder collects in the bottom of the furnace shaft proper, and accumulates until it reaches the holes G and L. It then overflows rapidly into the forehearth, carrying matte with it. In a short time, the level of the molten material rises above the top of the hole L, and from that time onward the blast in the furnace can no longer blow out through L, and is completely trapped. Owing to the pressure of blast, the level of molten matte and slag in the forehearth is several inches above that in the furnace proper. Eventually the slag-lip M is reached by the cinder, which then overflows quietly. Matte is tapped periodically from the tapping-notch O without stopping the furnace. Matte is never allowed to accumulate until it overflows at the slag-lip, the practice being to tap at stated intervals. The notch O is opened by a small steel bar, and pure matte, to the amount of about 1,000 pounds, is allowed to run off. During this operation, the level of the molten slag in the forehearth falls, but not sufficiently to admit of blast escaping through L.

By the simple insertion of a small clay stopper, the matte is stopped before cinder appears, thus avoiding all cinder picking. The whole process only occupies a few minutes, and is so perfect that for months a miss in tapping or closing up is not made. The large amount of molten slag and metal in the forehearth greatly facilitates a clean separation.

While using this forehearth in cold climates I have been troubled with its frequent cracking. On this account, I have designed a wrought-iron one of similar pattern, but with a separate, water-cooled slag-spout. This has been found entirely satisfactory and durable, and in spite of its greater first cost, is the more economical in the end. This is shown in Figs. 35, 36, 37 and 38.

Owing to the narrow water-cooled passage between furnace and forehearth, the Herreshoff furnace requires careful management to prevent the "sticking-up" of this notch. If only temporarily blocked by a fragment of coke or quartz, it can usually be cleared by probing it with a long rod of $\frac{1}{2}$-inch or $\frac{5}{8}$-inch iron passed through the slag-spout of the forehearth. In ordinary cases there is no occasion for emptying the forehearth for this purpose, but when the delay promises to exceed a very few minutes it is safer to tap the forehearth dry and even to empty the furnace through the little notch provided therefor.

The furnace can be blown in or out with great ease and little loss of time or coke, or it can be run simply on the day shift, shutting down nights without blowing out. To do this neatly, it is best to reserve all the foul slag made through the day, and after allowing the charge to sink tolerably low in the shaft, to charge the slag and a blank charge of coke, with a little fine coke on top,

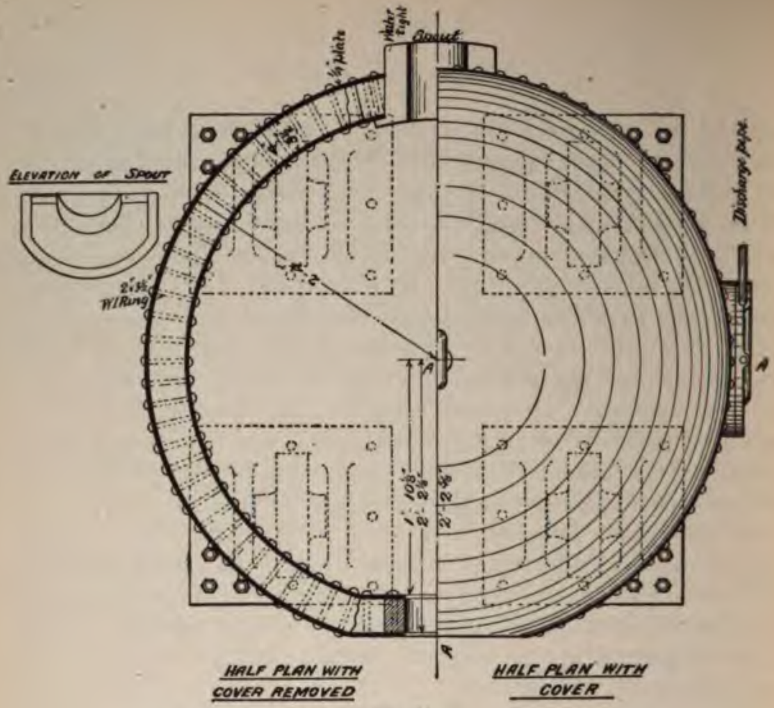

ELEVATION OF SPOUT
Spout
Water Tight
Discharge pipe
HALF PLAN WITH COVER REMOVED
HALF PLAN WITH COVER

FIG. 35.

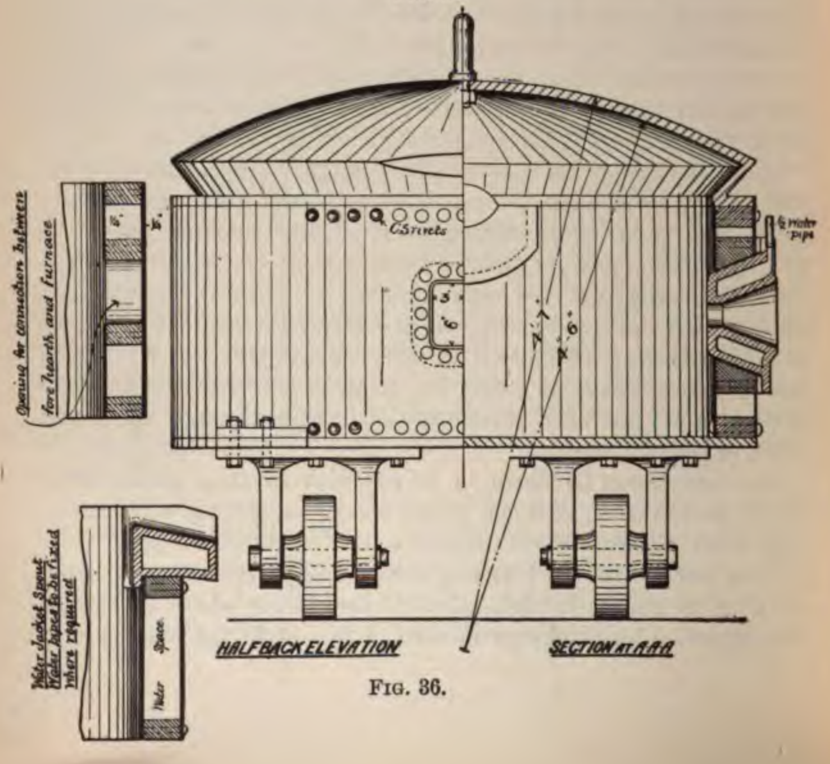
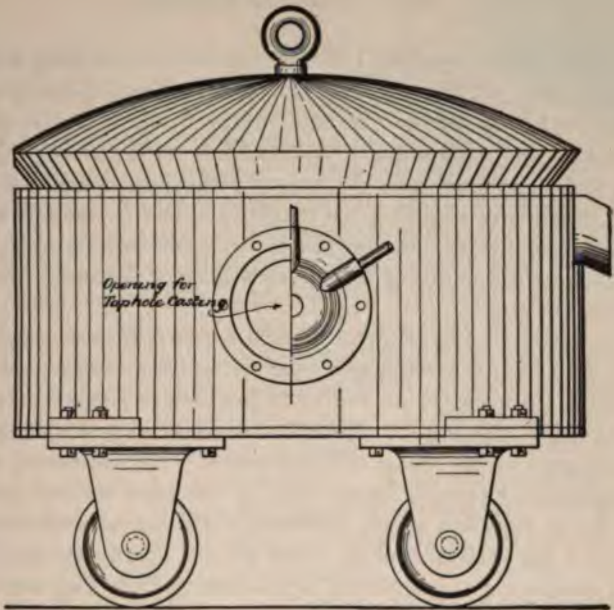
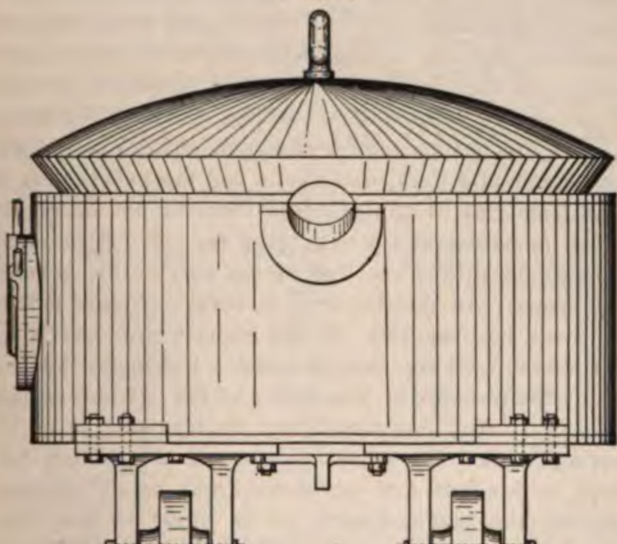

C.S. rivets
Opening for connection between fore hearth and furnace
Water Jacket Spout
Water pipes to be fixed where required
Water Space
HALF BACK ELEVATION
SECTION AT A.A.A.

FIG. 36.



SIDE ELEVATION

FIG. 37.



FRONT ELEVATION

FIG. 38.

and then, after tapping the forehearth and furnace completely dry, to plug all the tuyeres and every other opening where air could penetrate below. In the morning, the blast is put on at once, and the furnace shaft rapidly filled with the regular charge. Slag will commence running almost immediately and the smelting may, practically speaking, be taken up where it was left the night before, except that the first matte produced will be a little richer than usual on account of the slow roasting of the ore in the furnace during the night.

The Herreshoff type of forehearth is not suited to irregular running and frequent stoppages, nor to sudden changes in the ore mixture. Nor can it conveniently produce so siliceous a slag or so high grade a matte as a non-cooled hearth and one that permits easier access to the breast of the furnace. And above all, it cannot be managed by inexperienced men, as has been proved more than once by its rejection at points where it was actually the ideal apparatus for the circumstances. But for the steady smelting of uniform copper ores, producing a moderately free slag and a matte between 20 per cent. and 50 per cent., it can be run a greater number of hours in the year and with fewer repairs and less foul slag, than any other furnace with which I am acquainted. It also produces its matte absolutely free from slag or other impurities.

The separate tap-hole casting belonging to the Herreshoff forehearth is a circular, water-cooled bronze block. It is sometimes difficult to obtain these bronze castings perfectly free from flaws or blowholes, but the material is malleable enough to stand a good deal of caulking, and the life of even a bad tap-hole casting may be greatly prolonged by running its feed-water as hot as practicable. During the moment that matte is being tapped, a little more feed-water should be turned on, that steam may not be generated in the water space. As this tap-hole is always plugged as soon as a potful of metal has been run off, the furnace-man has to plug it against a tolerably strong, though small, jet of metal that is forced out both by the pressure of the blast and the column of metal in the forehearth. Any dampness about the clay plug or on the tap-hole casting is likely to cause a shower of the molten matte to blow back in his face, and as he naturally turns his head and shrinks from the bombardment, he is likely to miss his shot. Thus men are frequently burned, and matte is spilled about the floor. This little annoyance is avoided by a movable, swinging screen of sheet iron, having a vertical slot through it for the plugging-pole, and a pane of glass through which the furnace-man can see what he is doing. The main blast-pipe is also provided with a weighted clapper-valve, and, at the moment of plugging, the assistant pulls a wire which raises the valve and allows the blast to escape into the air for an instant. This relieves the pressure and makes the plugging much easier. Under ordinary circumstances, nothing larger than a carpenter's hammer is required to drive the tapping-bar through the clay plug. The 1½-inch tap-hole in the bronze casting is permanently plugged with clay, and through the center of this clay stopper, a one-half inch hole is left. This is the tap-hole proper, and is plugged with a fragment of plastic, slightly dried clay no larger than a cork. Outside of this minute plug a larger plug is forced in, and when the furnace is to be tapped, the attendant removes the outside clay with a small instrument like the blade of a pocket-knife. When the protected inner plug is reached, a light tap or two that might almost be given with the ball of the hand, is sufficient to drive the half-inch steel tapping-bar through the thin obstruction.

The following table gives a week's run of one of these furnaces, taken from the daily sheet of returns. It illustrates the steady running and the small amount of slag formed. As I have selected a week when there was no changing of forehearth, washing out of jacket, or important repairs of any description, the record is somewhat better than the average for the year would be.

During the week the furnace averaged 110 tons ore per 24 hours, making in the same time about 15 tons of 40 per cent. matte.

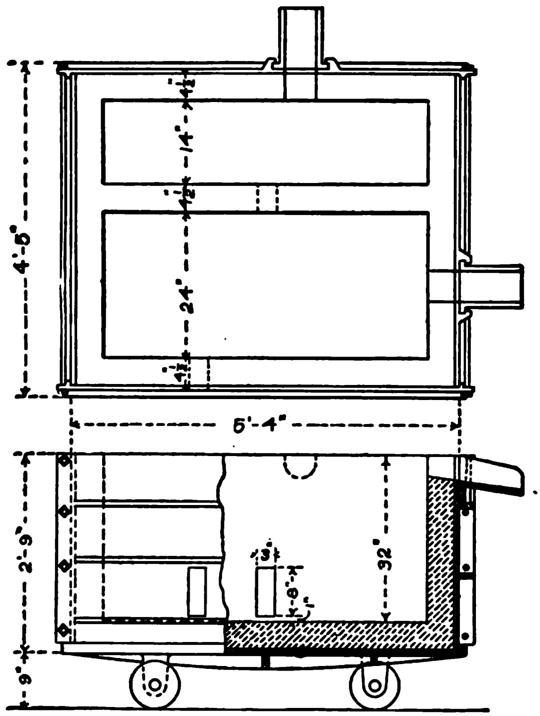
TABLE SHOWING ORE SMELTED, FOUL SLAG PRODUCED, AND DELAYS FOR ONE WEEK.

Date.	Ore Smelted. Tons.	Foul Slag. Pounds.	Blast off Furnace. Minutes.	Cause of Delay.
February 3	113	620	10	Clearing slag-hole.
" 4	109	1,100	25	Patching forehearth.
" 5	107	870	0	
" 6	103	400	15	Engine repairs.
" 7	119	1,220	20	Ore train late.
" 8	124	650	0	
" 9	96	700	15	Slag in tuyeres.
Totals......	771	5,560	85	

This makes the delays amount to 0.84 per cent. of the total time, and the production of foul slag equals about one-third of one per cent. of the ore smelted. It need hardly be said that the ores were exceedingly uniform and favorable, the plant excellent, and the furnace-men thoroughly experienced and interested in the results.

The Orford siphon-tap* forehearth is an outside settling device so arranged that the matte and slag are discharged from it in separate and continous streams. See Figs. 39 and 40.

It consists of a rectangular box, some 5 feet by 5 feet 6 inches,



FIGS. 39 AND 40.

formed of cast-iron plates strongly bolted together at the corners, and lined with a brick wall 4½ inches or 9 inches thick, according to the quality of the product. It is fastened firmly to the front of the furnace, just at the slag-run in the center panel, the lower middle portion of the anterior front wall of that structure forming

* This is an entire misnomer, as the apparatus here referred to, as used for the continuous discharge of the metallic product, has nothing about it pertain ing to the principles of the siphon.

its posterior boundary. It is divided longitudinally by a 9-inch wall of fire-brick into a greater and lesser portion, the area of the two compartments being about as 5 to 2, and the direction of the division wall bring parallel to the short axis of the furnace

The entire molten contents of the furnace discharge through a 2-inch by 4-inch opening (the slag-run) in the middle panel (the breast) into the larger of these two compartments, which is provided with a slag-spout, bolted to the upper edge of the front plate, while it communicates with the smaller compartment by means of a 3-inch by 8-inch vertical slot through the 9-inch division wall, about midway of its length and on a level with the floor of the forehearth. This smaller compartment also has a spout about 2 inches below the level of the spout belonging to the larger division, and on the outer side, instead of the end wall, for the sake of convenience.

A thorough understanding of this very simple and inexpensive contrivance will render it very easy to appreciate its management.

When the breast-hole is opened, and slag and metal first begin to flow, the larger compartment is soon filled, as the only means of communication between the two divisions of the forehearth is the closed slot in the lower part of the 9-inch division wall.

The molten products separate according to the law of gravity, and slag is allowed to flow through the spout of the large compartment until the drops of metal appearing show that it is filled with the more valuable product. The channel of communication is now opened by means of a crooked tapping-bar, and the metal flows rapidly through the same into the smaller compartment, until an equilibrium is established, and both divisions of the forehearth are partially filled with the matte, the communicating channel being far below the surface of the same, and consequently so situated that slag can never reach it unless it should sink below the metal, which is obviously impossible.

As the furnace constantly discharges its stream into the larger compartment, the forehearth is soon filled again, the metal sinking to the bottom and standing at the same level in both divisions, while the slag simply flows over the surface of the matte in the larger compartment.

As soon as the matte reaches the level of the spout attached to the small compartment, it begins to flow into a pot placed to receive it, and by judicious manipulation, and if a sufficient proportion of

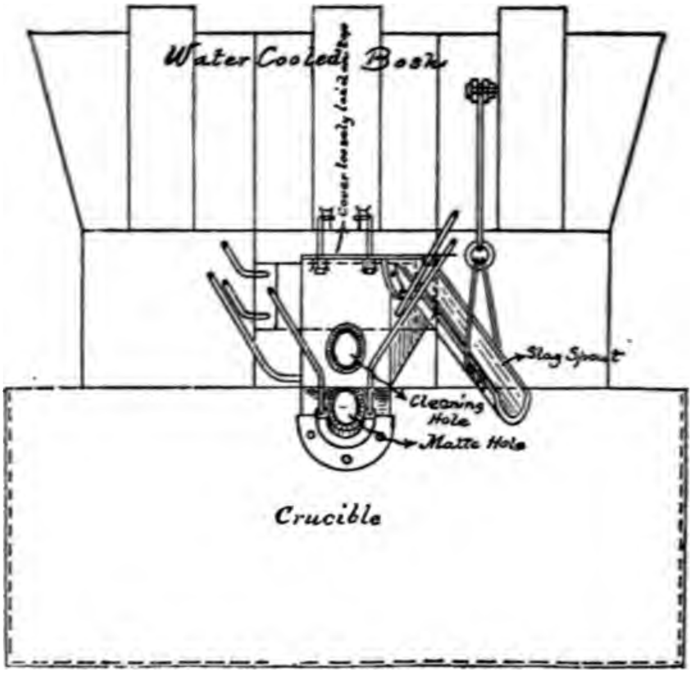


FIG. 41.—ELEVATION.

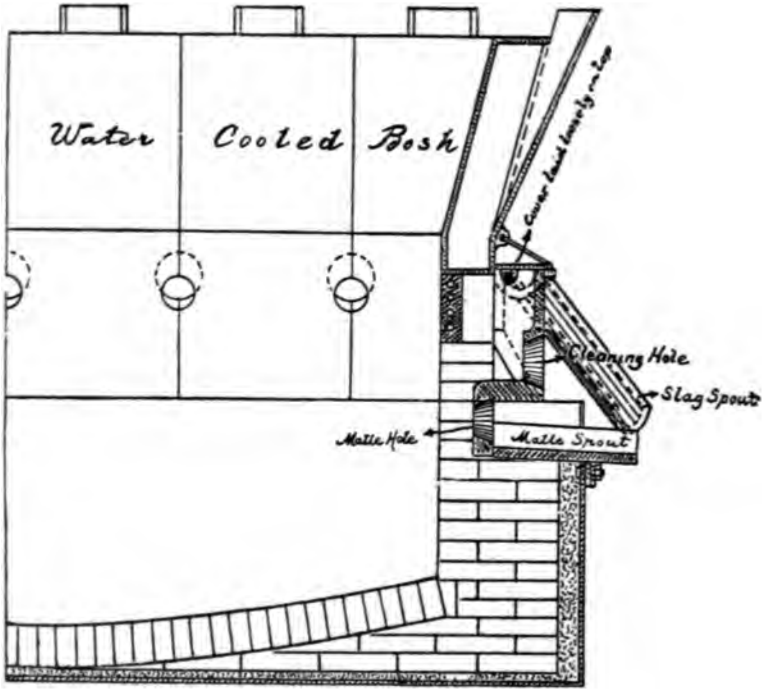


FIG. 42.—SECTION THROUGH VERTICAL.

matte is produced from the charge, a constant stream of each product may be kept running without difficulty.

The management of this siphon-tap requires considerable experience, as the matte stops occasionally without apparent cause, and requires a certain amount of manipulation and coaxing to keep running freely. This is accomplished by slightly damming up the slag-spout, which soon forces an excess of matte into the smaller compartment, or by clearing out the communicating orifice by means of a heated bar bent to the required curve.

With matte of 50 per cent. or over, the principal difficulty is found in the gradual filling up of the forehearth by chilling, while the matte containing 20 per cent. or less of copper, and produced in large quantities, has directly the opposite effect, thinning the fire-lining until the plates are endangered, and cutting away the division wall until the two compartments are virtually thrown into one.

But even under these circumstances, and as long as a vestige of the center wall remains, the separation of the matte and slag continues to be perfect, and by judicious repairing and nursing, a forehearth apparently in the last stage of ruin may yet do good service for many days.

An opening through the division wall 18 inches high by 24 inches wide, and actually involving two-thirds of the separating brick-work, is not incompatible with a perfect separation.

The larger compartment is provided with a tap-hole at its lowest boundary, and on the side opposite the matte division, and a large quantity of sand should always be at hand ready to make up into rough molds in case of any sudden necessity for tapping.

Mathewson's device (see Figs. 41 and 42) for separating matte and slag has usually been applied to lead-silver blast-furnaces where the matte is of very secondary importance. It may, however, prove useful to the copper smelter where exceptional circumstances demand the employment of an interior crucible, and where the amount of matte produced is very small and used primarily as a collector of the precious metals.

I have seen this apparatus doing most excellent work in Pueblo and elsewhere. The illustrations are taken from a paper by B. Sadtler, in *The Scientific Quarterly*, for June, 1893.

The matte is tapped from the lowest hole in the section, and should be free from slag. There is a cleaning hole above this, which is ordinarily closed. The slag flows out under a water-jacketed diaphragm, and through a spout which starts at nearly the level of the tuyeres.

*Reverberatory Forehearths.**—By a reverberatory forehearth, I mean an independent settling reservoir into which is discharged the molten material from the blast furnace, and which is heated from an independent source. This far, it has been found convenient to build this settler in the shape of a small reverberatory furnace.

To save time and repetitions, it will be advantageous to consider the reverberatory forehearth from the point of view of both the blast-furnace, and the converter departments.

Every metallurgist who is in the habit of running copper blast-furnaces at a rapid rate on tolerably uniform ores is aware that the larger portion of his delays, and outside costs and losses, are connected with the settling of the matte from the slag.

If he uses an inside crucible, he is likely to experience the train of evils already considered.

If, according to ordinary American practice, he employs an independent forehearth, he betters his condition decidedly, but is still frequently annoyed by the burning-out or chilling-up of the forehearth, the carelessness of the workmen in allowing matte to run over with the slag, and various other evils. These irregularities come largely from faults on the part of the furnace-men, and occur so much more frequently during the night shift, that I have been in the habit of saying, for instance, that under certain specified conditions, my forehearths would last for 20 night shifts or 40 day shifts.

The Bessemer-converter foreman may properly demand that his molten charges of matte shall be prepared for him:

(*a*) At the moment he is ready for them; and he may often need a double charge or charges for two or more converters in rapid succession.

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* While I have long believed in reverberatory forehearths, and have lately had opportunities to satisfy myself of their economy and effectiveness, I find that Dr. Iles, of Denver, has pursued the same subject with much more care and thoroughness than I have ever devoted to it, and has, indeed, patented a device of the kind. My present object in discussing this form of forehearth is simply to point out its possible value to copper metallurgists, and not to make any claims of either precedence or originality in the matter. At the Messrs. Elliott's Company's works in Wales, Christopher James is using reverberatory forehearths with advantage.

(*b*) So that the matte has a sufficiently high temperature to warm up a converter that has become too cool in the preceding blow.

(*c*) That the charging shall be accomplished quickly, else both the converter and the slowly trickling new charge may become unduly cold.

(*d*) He may desire to suspend using matte for a considerable time, and then require several charges almost simultaneously.

(*e*) Although it is a luxury he has never been much accustomed to, it would be highly advantageous if he could order his matte richer or poorer (within a 10 per cent. limit), according to the condition of the lining of his converters.

These demands, and various other causes, make it practically impossible to attempt to tap the matte directly from the ore blast-furnaces into the converters. As will be readily seen:

(*a*) One cannot always arrange to have a forehearth full of matte at just the moment that a converter requires a charge.

(*b*) It is impossible to change the ore mixture in the furnaces without disturbing the matte-ratio between furnaces and converters.

(*c*) The best managed blast-furnaces have their periods of depression and of exhilaration, which decidedly modify the amount of matte that they produce.

(*d*) The blast-furnaces must be run at a temperature considerably above that actually required to fuse the ore, in order to keep the matte in the forehearth sufficiently hot for the converters. The consequent waste of fuel is a steady and considerable expense.

(*e*) If there is any delay at the converters, it reacts directly upon the blast-furnaces, as they cannot dispose of their matte except by tapping it to one side and remelting it later.

(*f*) It diverts the blast-furnace foreman from his proper aim; which is to smelt as much ore as possible with the least fuel and the smallest losses. He has to constantly consider the needs of the converters, and unduly push, or hold back, his furnaces, which circumstance is ruinous to economical smelting, and also affords him an admirable and unanswerable excuse for any description of accident or bad work. It also destroys the spirit of rivalry and ambition which is so important a factor in large works.

(*g*) It causes endless complications and disputes between furnace and converter departments, as each is naturally looking out for its own interests with a total disregard for its neighbor's convenience or economy.

In view of all these drawbacks, experience has shown it to be more advantageous to go to the considerable delay and expense of breaking, transporting, and re-smelting the blast-furnace matte in a separate cupola that can devote its entire attention and interests to the needs of the converters. This naturally entails a heavy additional expense, amounting, in Montana, to something like $2.50 per ton of matte, besides requiring an investment for plant of at least $300 for each ton of matte melted per 24 hours.*

It is also a highly unreasonable and aggravating practice to deliberately cool matte that is all ready for the converters, and to resmelt it again with the consequent loss of labor, fuel, time, and metal.

Long before the days of bessemerizing copper, it had occurred to certain metallurgists that the separation of the matte and slag might be facilitated by heating these products in a separate reservoir outside of the blast-furnace, and by means of an independent fire. Since the almost universal adoption of independent forehearths, and especially since the development of the converter practice, the need for such an independently heated settling-reservoir has greatly increased, as may easily be gathered from the brief explanations just given. Without attempting to speak of the origin, history, or development of this idea, I will state my own views as to what seems to me the most convenient form of device for this purpose, and the chief advantages that may accrue from its use.

As the management of the reverberatory forehearth must be studied in conjunction with the running of the blast-furnace and converter-departments, so do its construction, maintenance, and repairs belong to the reverberatory section. It is simply a small reverberatory placed near the blast-furnace, and having the position of its fire-box changed to the side, instead of the end of the hearth; this modification is of course not essential, but usually seems more convenient. The blast furnace discharges its melted products into the hearth of the reverberatory through an opening in the rear wall of the latter, and the clean slag flows off continuously at the

* The Boston and Montana Company, at Great Falls, tap their matte into the converters, via an electrically-moved ladle, direct from the blast-furnaces and reverberatories. But few concerns have either the rich ores or the large capital necessary to arrange a plant satisfactorily on these lines.

front, or skimming-door end. The matte is tapped into the converters, either direct, or through the intervention of a ladle.

Very little need be said about the construction of this forehearth. The ordinary reverberatory furnace offers a perfect model, and the only changes required are those necessary to adapt it to its peculiar duties.

In planning its position in regard to the blast-furnace, the following points should be borne in mind:

1. To have it convenient for the removal of the slag.
2. To arrange it so that the matte can be tapped direct into any one of the converters (unless a ladle is used), and also, to have ample room to tap a very large charge of matte into sand beds at one side, and plenty of space to store 50 to 100 tons of matte in pigs.
3. To so place the forehearth that the breast of the blast-furnace can be easily and freely reached with tools.
4. To so plan it that the supply of coal for the forehearth can be economically and conveniently delivered and stowed.
5. To so plan the reverberatory stack, or down-take, that it may not be in the way and will not involve a too expensive construction of flues.

This little reverberatory should be constructed with a fire-box that can be completely closed, as in the long calciner shown in Fig. 22. This effects a considerable saving in fuel.

After the experience of Griffiths & James in Wales, and similar practice at Mansfeld and elsewhere, no one should think of using a sand, or quartz hearth in such a settling reverberatory. A slightly concave bottom of ordinary Stourbridge brick has already lasted two years in such a forehearth, running on very foul and leady mattes, and shows no signs of wear.

The only portion of the reverberatory that may require occasional looking after is where the surface of the slag touches the fettling. At this point it is liable to cut a groove all around the hearth, owing to its solvent action on the silica of the lining. Consequently, the hearth may require a little claying once in from three to ten days. By surrounding the hearth at this level with a 1½-inch pipe, using about 200 gallons of water per hour, I have almost entirely prevented the destruction of the lining. A sufficient crust of accretions is formed outside of the pipe to protect the walls very completely.

The size and depth of the hearth must depend upon the weight of the charge required for the converter, and the number of these vessels. In any case, a large body of matte kept constantly in the hearth maintains the latter at a uniform heat and acts as an excellent balance wheel for the entire process. Fifteen to thirty tons of matte are none too much, as there is no difficulty in constructing a hearth that can stand double that amount, providing it is properly built and ironed. The fire-box may be quite small, say 30 by 42 inches, as the amount of heat that is required in addition to that already provided by the molten products of the blast-furnace, is very small. When slag is *nearly* hot enough, a rise of temperature of a very few degrees makes an enormous difference in its physical condition, and may change it from a cold, red, sluggish, semi-viscid substance to a white, smoking, oily liquid, as thin as milk. Besides, the conditions here are totally different from those that prevail in a reverberatory smelting furnace. In the latter, the greater proportion of the fuel is consumed, not in actually melting the ore, but in

(*a*) Restoring the furnace to its normal temperature after it has been cooled off by skimming, tapping, fettling, charging, etc.

(*b*) Penetrating the feebly-conducting materials of the charge to reach the deeper layers.

(*c*) Raising half-molten masses from the bottom, where they often stick for a long time after the rest of the charge is ready to skim.

All these factors are absent in the reverberatory forehearth. It is never cooled off by charging, skimming, claying, or opening doors, excepting on the rare occasions when the hearth requires ten minutes' repairing. There is no non-conducting heap of ore to be penetrated by the heat, nor any half-fused masses sticking to the bottom, and there is a constant stream of white-hot matte and slag entering the forehearth.

It is quite practicable to make one forehearth serve for two or more blast-furnaces.

The advantages offered by some such form of reverberatory forehearth have already been foreshadowed in enumerating the drawbacks connected with the present system, which becomes particularly inconvenient when converters are employed. I recapitulate briefly.

The chief advantages that may be wholly or partially gained by the use of a reverberatory forehearth in works where the blast-furnace matte goes to Bessemer converters are:

1. The saving of the remelting-cupola operation.

2. The reduction of fuel in the blast-furnace to its lowest limits, as the ore requires no more heat than is sufficient to melt it so that it will run out of the furnace.

3. The complete escape from all the delays and costs connected with the chilling-up and burning-out of forehearths.

4. An increasing, rather than a diminishing, temperature as the slag flows through the settling device. This is, naturally, a most favorable circumstance for the separation of the matte. The settling is also favored by the constant presence of a large body of very hot matte in the forehearth.

5. The guarantee of any desired amount of matte for the converters at a moment's notice.

6. Permits irregular running, or even a complete stopping of the converters, without embarrassing the blast-furnace work; for it is as easy to tap the excess of matte into a sand bed as into the converters, and when the latter needs more matte than is furnished by the ore, the pigs can be slowly charged back direct into the reverberatory, and melted down without any extra fuel, their great fusibility and conductivity making this possible.

7. By keeping a stock of extra rich, and extra poor matte on hand, and charging the one or the other direct into the partly drained reverberatory, the grade of the converter charges can be rapidly varied.

SIZE AND SHAPE OF BLAST-FURNACES.

These important points are discussed in the chapter on "Pyritic Smelting," but I desire to supplement the same by a few words regarding the subject when considered from the standpoint of ordinary blast-furnace smelting.

With our present knowledge, it seems to me that blast-furnaces, whether water-cooled or of brick, fall naturally into two classes:

1. Blast-furnaces used simply for melting-down ores or other substances.

2. Blast-furnaces used for partial oxidation as well as melting.

1. Blast-furnaces used simply for melting-down ores or other substances.

But a small proportion of the copper blast-furnaces of the world fall strictly within this category. Typical examples of such furnaces may be found in ordinary foundry cupolas for the remelting of pig iron for castings, or in the cupolas for remelting matte for our copper Bessemer converters. These examples are particularly striking because the materials treated are free from gangue and from volatile constituents, and consequently yield (practically speaking) no slag, nor is their weight diminished, or their value increased, by the operation. The furnace process produces no chemical action in the charge. It merely changes the substances into a more convenient form for future treatment. As there is no chemical change in the ore, it follows that all the heat necessary for its fusion must be derived from coke, or other extraneous fuel. It is, therefore, a peculiarly wasteful and unsatisfactory operation, and after costing a considerable sum for labor, fuel, plant, time, and metal-losses, has not improved the actual condition of the substances treated by a single iota.

But there is a larger class of operations where better results are obtained by this same neutral, or reducing, system of smelting. This is where a certain proportion of the constituents of the ore are volatile, or, still more where they consist partly of oxides and silica (gangue). In the case of the volatile constituents, we remove these by merely melting the ore as already explained, and thus effect a certain slight concentration, the value of the product being in direct proportion to the amount of its volatile constituents. To take an extreme case: Suppose our ore to consist of pure iron pyrites without gangue, and carrying 10 ounces silver to the ton. Iron pyrites contains 53 per cent. sulphur, one-half of which is so loosely bound that it volatilizes as metallic sulphur at a moderate heat. Hence, 100 pounds of this ore would yield, on fusion, only 73½ pounds of product, that would contain silver at the rate of 13.6 ounces per ton. This illustrates a concentration effected by volatilizing certain of the valueless portions of the ore.

A concentration brought about by causing the already oxidized bases of the ore to unite with the silica present, is a much more common and more effective operation.

A typical example may be found in the Mansfeld practice, where the ore, as it comes to the furnace, contains nothing volatile, and the smelting operation is conducted in so powerfully reducing an atmosphere that none of the constituents of the ore can become oxidized in the furnace. The chemical action in the blast-furnace is here confined to the uniting of the silica with such bases as are already oxidized. But as the ore consists mainly of silica, magnesia, lime, and alumina, with a little oxidized iron, a high degree of concentration is obtained by this single reducing fusion, a 45 per cent. matte being produced from a 3 per cent. ore, while some 15 tons of slag go over the dump for each ton of product.

This is a unique case, for, as a rule, our ores contain so large a proportion of sulphur as sulphide of iron, or other sulphides, which will combine in the blast-furnace with already oxidized iron, and steal it from the slag, where it is needed, to carry it into the matte, and thus augment the quantity, and decrease the quality, of that product, that it is customary to roast the ore, by which process much of the sulphur burns away as sulphurous acid gas, and the iron is oxidized so that it can unite with the silica to form slag. We thus change a highly pyritous ore to a condition in which it somewhat resembles the Mansfeld ore. That is, we alter it so that it shall consist of a minute proportion of metal (sulphides), and an overwhelming amount of gangue rock (oxides), for iron, when oxidized, may be regarded as gangue. If this alteration is sufficiently thorough, our simple reducing smelting will bring about the desired result: *i.e.*, a small proportion of rich matte and a large proportion of fusible and poor slag.

But a calcination so thorough as to accomplish this result is expensive and not always practicable, for many ores contain too little sulphur to warrant roasting, while they have too much sulphur to yield a rich matte if simply melted down in a reducing cupola. Leaving out extra rich ores, it may be said that three-fourths of all the copper mines in the world are able by ordinary roasting and reducing smelting to produce a 30 per cent. or 35 per cent. matte from their average ores. But in the light of our present practice, this is an exceedingly inconvenient product. It lacks some 15 per cent. of being rich enough to send to the converters, while it is too rich to make it advantageous to crush and calcine it for a concentration smelting. It is the mission of the second division of blast-furnaces to add this lacking 15 per cent. of copper, without any additional operation.

Blast-furnaces that are used simply for melting, without any desire to oxidize the charge and thus enrich the matte, are characterized chiefly by the following features:

(*a*) Contraction toward the tuyeres (boshes).

(*b*) High ore column.

(*c*) Strong blast pressure (rapid smelting).

(*d*) Small, or moderate-sized tuyeres.

(*e*) Hot blast. (This is not a common adjunct, but would probably always be economical and effective for this peculiar class of work.)

I have spoken hitherto as though this simple reducing smelting were only in place under two conditions:

(*a*) For the mere object of changing the form of materials, as in melting pig iron for casting, or remelting matte, or rich ores, for the converters.

(*b*) For smelting ores that consist mainly of silica and bases in an oxidized condition (either naturally or by roasting).

I am strongly of the opinion that a third condition may soon be added to these, the success and economy of modern converter work having greatly changed the relation of the various metallurgical processes to each other.

At present, in America, we do not like to bessemerize matte that runs very much below 50 per cent. copper, 45 per cent. being the extreme limit for regular work. It would be considered ridiculous to attempt to bessemerize a matte containing only 20 per cent. or even 15 per cent. copper. There are three main difficulties in the way of effecting this exceedingly desirable object:

1. Converter linings become too rapidly destroyed by mattes below 45 per cent. copper, and no basic, or artificially cooled, lining has yet been a success, nor have we been able to induce the ferrous oxide produced from the matte to content itself with artificially supplied silica instead of robbing it from the lining.

2. Slag is made too rapidly when the matte contains much iron, and no method for its continuous removal from the converter has yet been successful.

3. The amount of copper, or of rich matte, derived from a very low grade matte is too small to manipulate without some continuous method of introducing fresh matte.

If these difficulties were obviated, and none of them appear insuperable, it seems to me that where coal is cheap and coke dear, as in many places in the West; or where water-power is available, as at Great Falls, Montana, our simplest and most economical way of handling such ores as those of Butte (or of most other American copper, and copper-silver-gold districts), will be to smelt them raw in large blast-furnaces with coke and a hot blast, creating a powerful reducing action, and running the low-grade matte continuously into Bessemer converters, where it will be blown up to a point when the resulting slag becomes rich enough to require resmelting, (which, with reverberatory settlers may be 60 per cent. or more). This matte, tapped, or run direct into the finishing converters, will yield a very small amount of slag for re-treatment, the operation being so regulated that there will be just enough converter slag to flux the highly siliceous raw ore in the blast-furnaces. I would propose to greatly contract the present processes of mechanical concentration at Butte, and a very small proportion of the copper thus rescued from loss would pay for the extra coke required to smelt the raw ore. The ore slags might easily run from 45 per cent. to 50 per cent. silica, and would be specifically very light, and contain under 0.3 per cent. of copper. This would greatly simplify and cheapen the entire metallurgical plant and treatment, and, in the instance specified, would largely substitute the power of the Missouri River for hand labor and fuel. It would abolish the crushing and roasting of the ore and curtail the process of mechanical concentration by some 60 per cent. or more.

The Butte metallurgists have faced and solved problems considerably more difficult than this one appears to be. The bessemerizing of matte containing 20 per cent., and less, of copper is an accomplished fact in France and Russia, though I have not myself seen it, nor do letters to me from metallurgists engaged in the work give me any satisfactory practical reasons of how they induce linings to stand under such circumstances. As regards the changing of the converter process from an intermittent, to a continuous, operation, I cannot see that any insuperable obstacle exists.

2. *Blast-furnaces for partial oxidation as well as for melting.*

This section comprises by far the greater portion of the copper blast-furnaces of this, and other countries. The operation varies from a slight, and often unsuspected, oxidation of a little of the sulphur and iron of the charge when smelting ordinary raw or roasted ore, to the most pronounced form of pyritic smelting.

As this latter process is considered fully in a separate chapter, I must confine myself, in this section, to furnaces where no especial attempt is made to utilize the ore itself as fuel, or, in other words, to practise pyritic smelting.

The extent to which oxidation shall be pushed in the blast-furnace is a point that has a most important bearing on the economy of the entire process, and one that demands for its correct decision the greatest experience and judgment on the part of the metallurgist. Each case has to be judged upon its own merits; but under the great majority of conditions and with the present general arrangement and construction of plants, it will be found decidedly advantageous to use the blast-furnace as a partial oxidizer, and to produce a richer matte than would naturally result if the charge were simply melted down in a reducing smelting, as occurs in the crucible assay for determining the amount of matte that will be produced by a given mixture.

No one need shrink from this practice as a dangerous or untried experiment. Probably the very metallurgist who would refuse to listen to a suggestion to use his blast-furnace as a partial roaster or calciner, is actually running it more or less on these lines without ever having realized the fact. If he doubt the truth of this statement, let him merely decrease the size of his hearth and of the shaft slightly above the tuyeres, use smaller tuyeres, and thinner layers of charge, and a stronger blast. Then, when he observes his matte increase in quantity and decrease in quality, and his slag become siliceous from the robbery of its iron by the unoxidized sulphur, he will realize that he has been partially calcining his ore in his blast-furnace, and has been practising what I term "Compromise Pyritic Smelting."

It is frequently a matter of the greatest value to employ this partial oxidation of the charge in the blast-furnace, and it is always useful to feel that one at least knows how to accomplish it if occasion should require it.

The difference between this method and the rapid process of merely *melting* the ore, which was considered in the previous section, lies entirely in so running the furnace that a partial oxidizing atmosphere is substituted for the powerful reducing atmosphere that characterizes the other operation. This is effected mainly by diffusing the heat over a greater area and lessening the sudden violence of the combustion at the tuyeres. When we wished to simply *melt* the ore in the most rapid manner possible, we constricted the shaft at the tuyeres and blew a strong blast into this concentrated mass of coke and ore, producing a very high local temperature and a dense atmosphere of carbonic oxide gas. The ore melted almost instantaneously and dropped into the neutral hearth below. The sinking of the charge was rapid, the heat was concentrated in the tuyere zone, and the ore had scarcely reached a red heat before it was fused and removed from all chemical influences.

To obtain a certain amount of oxidizing effect, we need pretty much the opposite set of conditions, and the mere enumeration of one or two of them suggests, or rather compels the remainder. We need a light blast; but a light blast cannot penetrate a thick column of charge, nor will it give any reasonable capacity for the furnace. We are forced, therefore, to use a furnace which is narrow in one of its dimensions, so that the blast can penetrate the ore column, and we must lengthen it in the other direction in order to obtain sufficient capacity. This brings us to the long, narrow rectangle as the only suitable form for our purpose, and furnaces are now constructed with a shaft up to 14 feet in length, the ordinary width being 32 inches to 38 inches. We desire to avoid the concentration of heat and the reducing effect inseparable from a contraction of the shaft at the tuyeres, and find that we obtain the best oxidizing results from perfectly perpendicular walls. As the blast pressure must be light, we make up our deficiency in oxygen by increased volume of wind, and consequently are obliged to enlarge the diameter of our tuyeres to 4 inches and even 6 inches. A high ore column strongly favors reduction. Hence, we employ an ore column only high enough to utilize the heat as far as practicable, and to give the ore time for partial oxidation during its descent. Four to six feet from tuyeres to charge door is the average height. We expect our charge to be moderately hot on top, as the furnace is acting as a roaster to the very tunnel-head. We prefer a cold blast, as heated wind leads to the concentration of temperature and rapid smelting that we are trying to avoid.

To obtain the large volume and low pressure of blast that we require, a fan blower may quite possibly be the most effective and economical machine that we can employ. Its main disadvantages are its high speed, small pulleys, and large belts.

VARIOUS OPERATIONS ABOUT THE BLAST-FURNACE.

The charging of the blast-furnace by shovel is being gradually replaced by more or less perfect mechanical devices. Where hand labor is still employed, the ore and coke should flow from bins direct into two-wheeled charging barrows, that can be dumped upon the cast-iron floor at the charging door of the furnace. Scoop shovels should be used in charging both ore and coke, and no man who finds them too heavy can make a rapid feeder. The railroad dump-cars will, of course, run directly over the charging bins, and drop their contents into the latter. Where the lay of the ground is unsuitable for a terraced construction, an inclined plane with winding-engine to haul the railroad cars up over the bins is much more economical than any form of elevator, and much better suited to handling large quantities of material without confusion. All ore, coke, and slag should be delivered in this manner, and wheelbarrows should be regarded with suspicion.

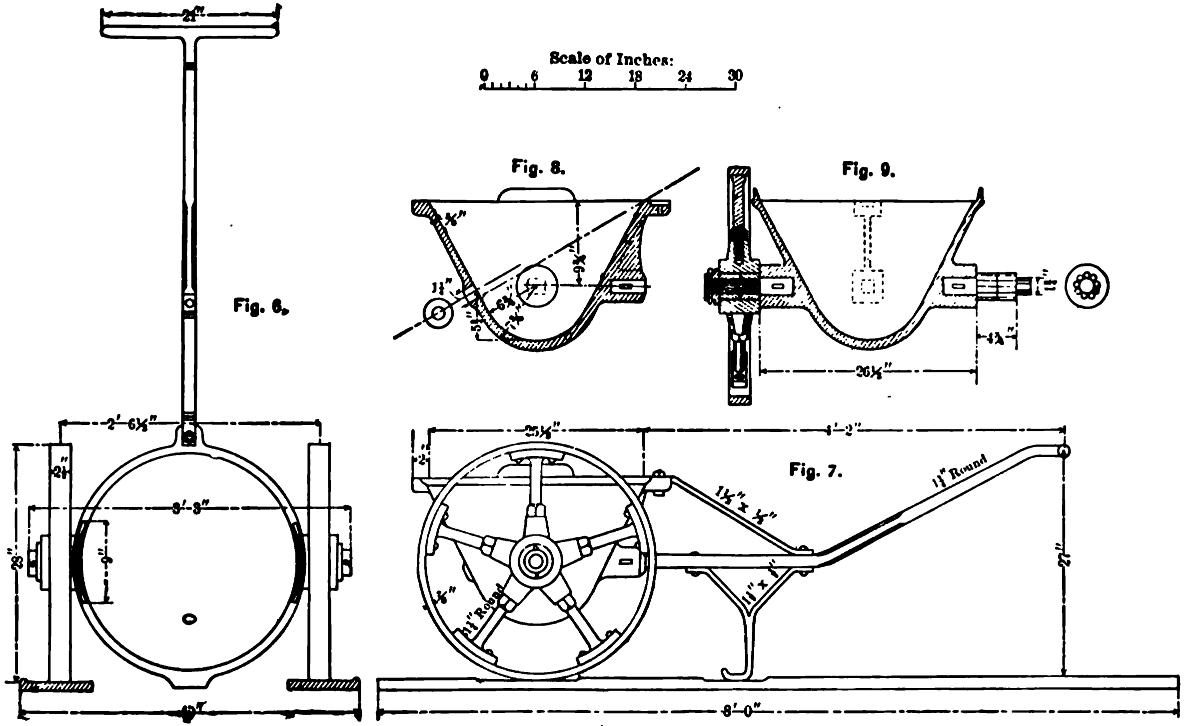
It is cheaper and more convenient to dump fuel and ore into the furnace direct from the charging barrows. To do this to advantage, it is necessary to construct the blast-furnace with a side flue through which the gases are drawn off below the level of the charging door. This is arranged in the same manner as with lead furnaces, by a thimble introduced into the upper portion of the shaft, the gases being drawn off from the annular space between the thimble and the furnace walls. By this device, the bulky housings and overhead flue are abolished, and the furnace opening consists merely of a rectangular slot in the unencumbered charging floor.

The Pueblo Smelting Company has adapted an excellent device whereby the filling of the furnaces is accomplished by means of a long, narrow charging-car corresponding to the rectangular opening of the furnace-top, and running on a track that straddles, and is at right angles to the long axis of all the blast-furnace tunnel-heads. By a simple stop-mechanism, the attendant controls the car so that it shall deliver its load of coke and ore into any furnace requiring it. A single man on the charging floor can thus attend to the charging of six large furnaces.

So far from finding it derange the running of the furnace, I have obtained better and more uniform results the nearer I have approached to strictly automatic charging. If one corner of the furnace threatens to chill, it is easy to arrange the mechanical device so the ore shall be diverted from the chilled portion for a charge or two, and the substitution therefor of a few hundred-weight of basic slag, and the plugging of the one or two tuyeres that are involved in the chill, will soon set matters right again.

The handling of the products of the blast-furnace has also been considerably cheapened of late years.

The matte is either tapped off at intervals into slag-pots, only about 1,000 pounds being drawn off at each tapping, in order that the matte in the forehearth may not be unduly lowered, or it is tapped in large charges direct into the converters, or converter-ladles, or it may be tapped in considerable amounts into sand beds or iron (or soft steel) molds. The latter method is generally used where matte is to be shipped or sold, as it gives a cleaner product and lessens the chance of irregularities in the sampling. It can also easily be shotted by a strong jet of water, though this makes



SLAG-POTS. - 1884 TO 1892. - NO. 2.

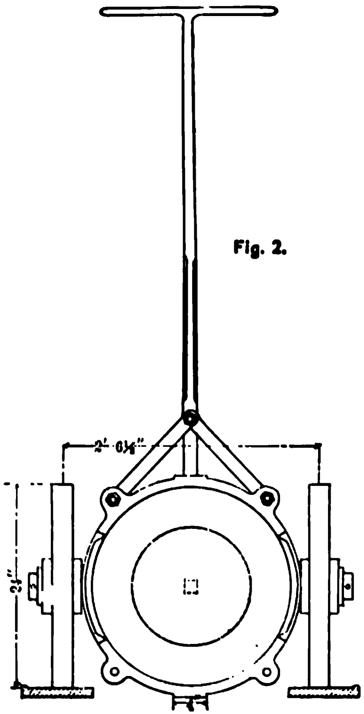


Fig. 2.

Fig. 4.

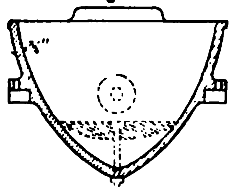
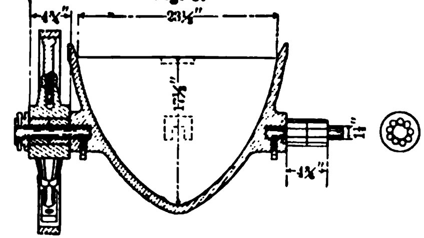


Fig. 5.



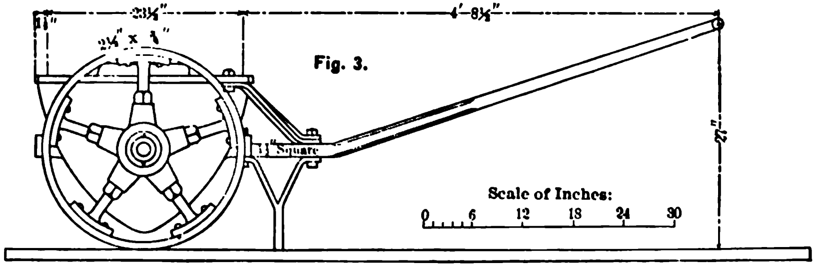


Fig. 3.

SLAG-POTS. - 1884 TO 1892. - NO. 1.

many polished, bean-like granules that seem scarcely worth the trouble of crushing, and yet resist the action of the calcining furnace.

Slag may be handled

1. In small pots by man power.
2. In large pots, or other vessels, by mule, or steam power.
3. In mechanical pan conveyers.
4. By granulation by water.

1. In small slag-pots. Although these useful little pots are being rapidly superseded by more economical devices, they still retain their place at many good works, and are worthy of careful consideration.

Mr. H. A. Keller, in an excellent paper on the subject,* gives some interesting cuts of slag-pots, which I copy, together with his description and comments.

"In the accompanying illustrations, Figs. 2 to 5 inclusive represent the cart now in use at the Parrot works, and Figs. 6 to 9 a cart similar to the one introduced by the writer at the Philadelphia works at Pueblo, Colorado.† Parts of these pots have been in use for a number of years, while other parts are of more recent date.

The cast-iron track shown in the drawings is laid into that part of the slag-dump which by constant usage is apt to become specially rough and uneven. A rough dump, besides adding to the work of the slag-wheeler, greatly increases the necessary repairs. Further away from the furnaces the dump is leveled by "slag squares" or slabs of slag formed by pouring, which are constantly kept up to its edge. These are best made 2 feet by 4 feet and from 8 to 12 inches deep. After a mold of these dimensions has been formed by means of rails or cast-iron plates and cold slag, it is partially filled with large pieces of cold slag, which are then cemented together with liquid slag poured simultaneously from several pots.

The slag-cart consists of three parts, the bowl or pot proper, the wheels, and the handle or foot.

There are two styles of bowls now in general use. These are represented in the accompanying Figures, as No. 1 and No. 2. On account of its straight sloping sides, the pointed bowl, No. 2, allows the matte to settle more readily. It is therefore preferred

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* *Transactions American Institute Mining Engineers*, Vol. XXII., p. 574.

† Hofman's *Metallurgy of Lead*, p. 203.

when but little matte is suspended in the liquid slag, which matte is to be saved in shell and bottom. To avoid unnecessary disturbance, this bowl is provided with a 1¼-inch hole, through which the liquid slag is tapped by a ⅞-inch bar, usually of hexagonal steel. Such a tap-hole was first used in this country by Mr. W. B. Devereaux, at Aspen, Colorado.* Its location varies, of course, with circumstances. After much experimenting, the writer, for instance, when employed at the Philadelphia works, determined to locate it as shown in Fig 8.

The bowl No. 1, with rounded sides, has the advantage of greater capacity than one of similar dimensions with straight sides. Such a bowl is therefore preferable when it is intended to dump out the entire cone. A more shallow bowl (shown in Fig. 1), introduced by Mr. A. Eilers and universally used in early Leadville smelting practice, is gradually disappearing with the increased size of slag-dumps, since it does not permit as great capacity as those shown in Figs. 2 to 9.

Bowl No. 1 requires the axles to be fastened to it with set-screws, while the straight sides of No. 2 leave room for securing the axles with wedges. In the latter case, each axle is provided with a square stub ¼-inch larger than the diameter of the axle. The axles of the third form of bowl (Fig. 1), are carried by one continuous piece of square iron, bent to conform with its shape. This is fastened to the solidly cast side-lugs by stirrup-clamps and further held in place by passing between two guiding-lugs at bottom of bowl.

The lug shown at the rim of each bowl is a great protection to the spot where dumping causes most wear. It was first suggested by Mr. C. T. Limberg of Leadville, and is now universally used.

The splash-guards prevent the liquid slag from spilling upon the hubs. They were, I believe, first introduced at the Grant works, Denver, Colorado.

The false bottom for pot No. 1, shown in dotted lines in Figs. 2 and 4, has been in use for several years at the Parrot works. It consists of a cast-iron disc held in place by a countersunk ¾-inch bolt. If the original bottom is very badly damaged, a washer may have to be used besides. Though such an arrangement does not give satisfaction with a heavy flush, it behaves admirably with slag or matte running slowly. With matte, it has the additional ad-

* See Kerl's *Metallhüttenkunde*. 1881, p. 100.

vantage of producing a flattened cone which is more easily broken than a tapering one. New bowls are used for slag at the Parrot, and last about eighteen months. After that, being provided with these false bottoms, they last almost as long on matte. Another contrivance for using a bowl after its point is worn out has been described by Mr. R. H. Terhune.*

By the device shown in No. 1, the bowl is made reversible, the cart being at the same time steadied by fastening the handle with two straps instead of one.

The style of wheel represented† consists of a cast-iron hub, with wrought-iron spokes and tire. It is mounted upon a steel roller-bearing. The hub is tapped to receive the end of the spoke, which for that purpose is threaded. For further tightening, each spoke is provided with a jam-nut. After the spokes have been carefully adjusted, the wrought-iron tire is shrunk upon their outer ends and is subsequently fastened to them by means of countersunk rivets. This tire being the part of the wheel subjected to most wear must be made sufficiently heavy and strong without being clumsy. The advantages of a wrought over a cast tire are evident, particularly when, as in this case, the former is well fastened and easily repaired. The anti-friction rollers require no oiling, or at least but little. A double set of these rollers is put in loosely around each axle. Thus arranged, they are prevented from wearing so as to cause appreciable friction, *i. e.*, by running upon one another. By using $\frac{3}{4}$-inch rollers, a $1\frac{1}{4}$-inch axle takes two sets of nine; and for each $\frac{1}{4}$-inch increase of axle-diameter an additional roller is required.

The foot or handle is practically the same for all forms of slag-cart. The essential point is that it shall be of sufficient length, and its crosspiece wide enough for convenient pushing. It is fully illustrated in the drawings and needs no further comment.

All rivets and bolts in either No. 1 or No 2 are $\frac{5}{8}$-inch in diameter. The average weight of No. 1 is 563 pounds. Many matte-cones taken from such pots with false bottoms gave:

631 pounds for 55 per cent. copper.

$603\frac{1}{2}$ pounds for 52 per cent. copper.

597 pounds for 47 per cent. copper.

$574\frac{1}{2}$ pounds for 44 per cent. copper.

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* *Trans.*, XV. 92.

† Patented by Cole, Gaylord and Keller.

A large number of slag-cones taken from pots with original bottoms gave an average of 496 pounds. Their composition was:

SiO_2	38
FeO	46.5
Al_2O_3	10.5
CaO	3
Total	98

The copper contents are from 0.3 to 0.4 per cent., present either as Cu_2O or as suspended matte.

It may be of interest to mention here that a large number of copper-matte analyses from reverberatory and blast-furnaces, comprising the different grades and extending over several years, gave a constant tenor of from 21 to 23 per cent. sulphur. Many of these analyses showed the presence of magnetic iron, sometimes in considerable quantity. Copper-mattes would accordingly correspond to either of these formulæ:*

$$(Cu_2S)_x + (Fe_2S)_y \qquad \text{or}$$
$$(Cu_2S)_x + (FeS)_y + (Fe_3O_4)_z.$$

No. 1 slag-pot casts, as a rule, six full slag-bricks. A great many bricks, being weighed, gave an average weight of 54 pounds each. The slag given above weighs therefore 216 pounds per cubic foot. Taking the same quantity of water at 62½ pounds, gives a specific gravity of 3.47. These seem fully as accurate as similar determinations made in many laboratories of Western smelting works, most of which are necessarily crude in this line of research."

The foregoing remarks indicate the chief points that require attention in the construction of slag-pots. If they are properly made, there is one certain way to make them last. This is to have plenty of them, so that they may have a chance to cool before being used again. The damage effected by constant overheating is far beyond the mere burning of the iron or distortion of the pot. The main injury arises from the slag or matte "welding" to the overheated pot and requiring much hard sledging on the exterior before the slag cone will drop out of it. It is this treatment that destroys pots.

Another point in regard to the economical handling of slag in these pots is the importance of extreme neatness about the dump,

* The writer is indebted to Dr. Edward Keller for these formulæ.

furnace, and runway. An old pot-hauler, who understands his business, will keep his floor and runway so free from splashes of slag that he can push his pot with the very slightest expenditure of power. He keeps three pots always at the furnace; a pot that has just been filled and is waiting until a thin skin has formed on the surface to prevent splashing; a pot that is being filled; and a cold, empty pot to use next. When the pot at the furnace is about five-sixths full, the furnace-man holds a ladle under the slag stream, while the pot-hauler removes the full pot and shoves in the empty one. The full pot is allowed to stand at one side, in order to chill on the surface, while the one that is already skimmed over on top is pushed out on to the dump. This skimming over of the surface prevents all splashing of liquid slag, and a pot-hauler who adopts all these little precautions has not only an easy track to push on, but has almost no sweeping-up to do.

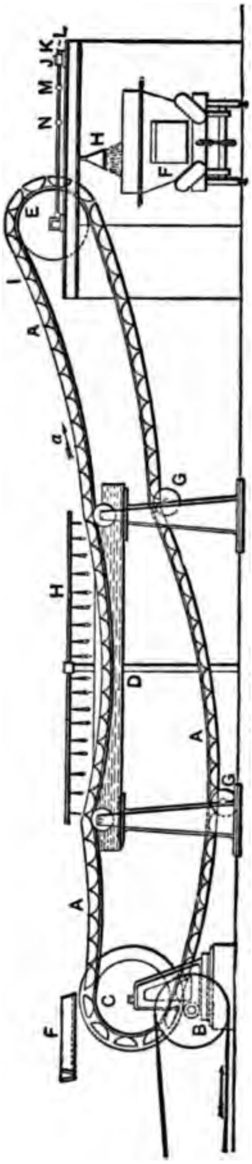
It is equally important to keep the brink of the dump in good repair, and with a sharp edge and steep slope. When two or more furnaces are running, a dump-man is required on both day and night shifts, and can save his wages several times over.

2. In large pots or vessels. Dumps have grown so large, and furnaces smelt so much more ore than formerly, that it has been found convenient to sling two or more large pots on a frame running on a track, and use a mule to drag them to the edge of the dump. The pots may be so hung as to be easily tipped, or their liquid contents may be tapped through a hole near the bottom. Large frames are also used, which stand upon an iron car, and taper a little toward the top. When the enclosed block of slag has chilled sufficiently, the frame is hoisted off, and the carriage is inclined by suitable mechanism so that the block of slag slides off, and down the face of the dump. These latter carriages may be hauled by mules, or coupled into a train and shifted by a small locomotive. Their construction and manipulation is a familiar part of the metallurgy of iron.

3. Mechanical pan-conveyers. These devices also have been principally developed in the metallurgy of iron, to whose textbooks the student is referred. One of the most convenient that I know is an English patent called "Hawdon's Slag Carrier." It is plainly shown in Fig. 43.

4. Granulation by water. After trying various shaped jets and other more or less elaborate devices, the majority of the smelters now granulating their slag by water have come down to a simple

stream of that liquid, running through a narrow trough with con-



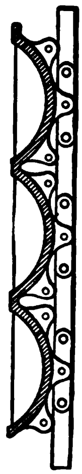


Fig. 43.

siderable velocity, and into which the stream of slag drops. Very little steam or noise is made, and the practice is entirely satisfactory providing there is enough water to thoroughly granulate and remove the slag, and prevent any possible formation of a solid cone with liquid contents, which might cause a very serious explosion. I know of no accurate figures as to amount and pressure of water required under various conditions; but I have found that a stream of water delivered by a two-inch pipe under a head of 12 feet, and flowing through a launder with a fall of one inch to the foot, will thoroughly granulate and remove 100 tons of heavy, ferruginous slag per 24 hours.

One of the main difficulties by this system is mechanical, the destruction of the launders by the granulated slag. Cast-iron plates are generally used as bottoms, but I have been able to obviate the expense of constant renewals by forming the main launder of slag-brick. The bottom will last a long time and can be quickly repaved.

The water can, of course, be used over and over, a waste of 5 per cent. being experienced at each time.

Where there is not sufficient fall to permit of the direct discharge of the slag over the dump, a bucket elevator makes an ideal arrangement for elevating the granules to any desired height, and thus building the dump up in the air. As this granulated slag makes excellent material for roads and embankments, the sluice may discharge direct into the railroad cars, the water leaking through the sides of the car or flowing over the top. A ditch at the lower side of the track will catch all the water and lead it to a suitable reservoir.

It is obvious that, when granulating the slag by water, a careful watch must be kept on the level of the matte in the forehearth. An excellent control can be kept on the furnace-man by examining the little mound at the foot of the sluice, as any matte granules will be found here in a concentrated form. Besides repeated examinations during the day, this mound should always be carefully panned at the change of shift, else each furnace-man will claim that any matte found therein was made in the other shift.

CHAPTER XII.

BLAST-FURNACES CONSTRUCTED OF BRICK.

ALTHOUGH some 90 per cent. of the copper blast-furnaces in the United States are now water-jacketed, one progressive and thoroughly experienced concern, The Orford Copper Company, still uses the large brick Raschette furnaces that they introduced some 20 years ago. Such results as they obtain in these furnaces, which are now cooled in the region of the tuyere openings by means of water circulating in pipes embedded in the brick work, cannot properly be ignored. This type of furnace also demands so much care and skill in its management, that it forms a peculiarly instructive study.

The distinctive peculiarities of the "Orford" furnace, as this altered and improved form of Raschette furnace is usually designated, aside from its unusual size, are the large number and diameter of its tuyere openings—14 of 6 inches diameter; the absence of any interior crucible or space for the collection of the fused products; the substitution therefor of an exterior forehearth or basin, and the construction of the latter in such a manner that two continuous streams—of slag and metal respectively—flow therefrom into ordinary slag-pots, without any blowing through of the blast, or delay for tapping and other related manipulations.* The latter arrangement may be applied to any furnace of sufficient size, it being absolutely essential, for the prevention of chilling, that a large quantity of molten material should constantly traverse it. If the product is a matte of high grade, 60 per cent., and over, a much larger quantity is necessary to prevent chilling than if the metal is of poorer quality. The rapid chilling of the former is due not to its possessing a higher fusion point, but because its capacity as a conductor of heat increases with its percentage of copper.

When the smelting mixture is exceedingly rich, so that a very large amount of the copper-bearing product results, it is even pos-

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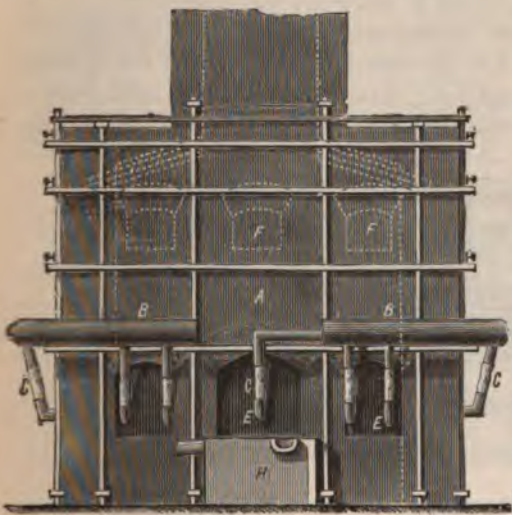
* See section on "Forehearths" for detailed description of this device.

sible, by rapid smelting, to maintain a constant stream of metallic copper—a practice that may be regarded as a curiosity rather than as ordinarily feasible.

A detailed description of the construction and subsequent management of this form of furnace will bring forward the points already referred to, and illustrate the practice that up to the present time has been found most advantageous, and which has cheapened the smelting of copper ores to a remarkable extent.

The outside measurement of the furnace being 8 feet 5 inches by 16 feet 8 inches, an excavation should be made at the intended



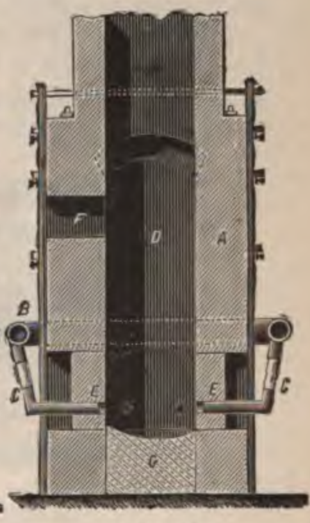


FIG. 44. ORFORD BRICK FURNACE. FIG. 45.

site some three feet larger in every direction than the figures just given, and of sufficient depth to reach solid ground and insure a proper foundation. A depth of 4 or 5 feet will usually suffice, the pit being immediately filled with concrete; or, where possible, the pit should be filled to nearly the surface with molten slag.

The walls of the furnace should be begun a foot below the ground level, and should consist entirely of fire-brick up to the tuyere level, where the panels shown in the cut are begun. Up to this point, the walls are 30 inches thick, of solid fire-brick, while the panels are only 18 inches thick, thus being more accessible for repairs, and containing the tuyere openings. The rear wall is divided into three panels equally spaced, and supported on each side by the full thickness of the wall, forming columns at each corner, and between the weaker portions, that are chiefly relied upon to carry the weight of the superincumbent structure. The panels are 30 inches wide and 33 inches high, and are strongly arched over with three rows of fire-brick, above which the full thickness of the wall (30 inches) is maintained to the top of the structure. Each panel is pierced by two 6-inch square tuyere-holes, equally spaced, excepting the central front panel, which contains only a small orifice for the slag-run, at a point some 10 inches below the tuyere level. The panel referred to forms the breast of the furnace, and is not closed in until the last moment.

The total number of tuyere openings is 14—6 behind, 4 in front, and 2 at each end. The interior rectangle is 3 feet 5 inches wide and 11 feet 8 inches long, although any exact adherence to these measurements is unnecessary, the interior of the furnace

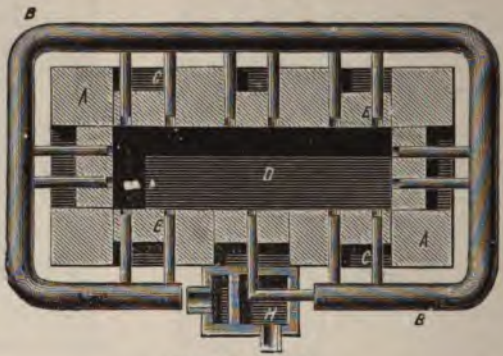


FIG. 46.—PLAN.

being soon burned out into an irregular shape and usually much larger than the size just given.

Strong tie-rods, provided at their extremities with loops, and buried deeply in the foundation, are placed in position as indicated in the cut. Unless the transverse rods can be placed at a depth of two or three feet below the surface, they should merely be fastened into the wall by hooks, as they would certainly be melted away in time.

The brick should be laid with the closest possible joints, and in a very thin mortar made of half each of raw and burned fire-clay, ground exceedingly fine.

Heavy railroad iron may be used for binders, and should be used rather more than less liberally than shown in the illustration, as

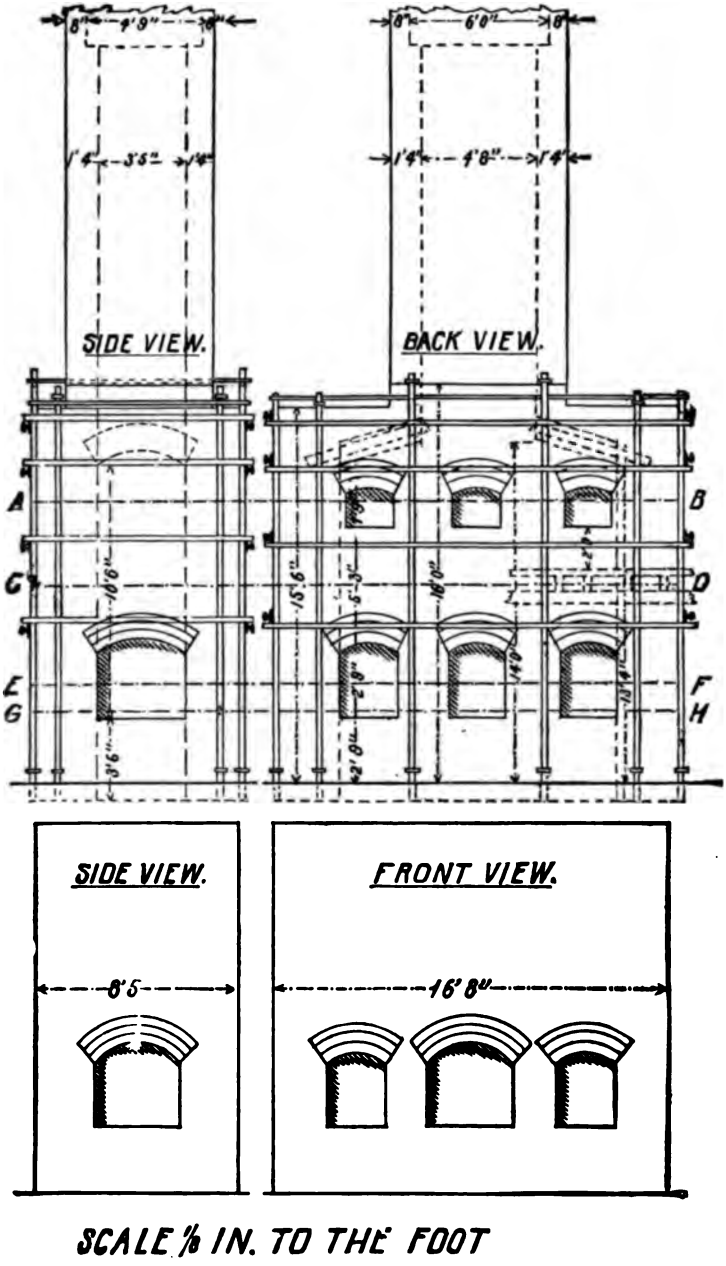


FIG. 47.—THE ORFORD BRICK FURNACE.

the expansive force is enormous when the furnace is in full heat, and any serious cracking tends greatly to shorten its existence.

If fire-brick are expensive, the outside lining, above the panels, and to a depth of 12 inches, may be constructed of red brick, although this is not recommended.

The usual height from the tuyeres to the threshold of the charging-door is 8 feet; but this, of course, may be varied to suit the character of the ore to be smelted. The charging-doors are three in number and of large size. All further details of construction are plainly shown in the cut.

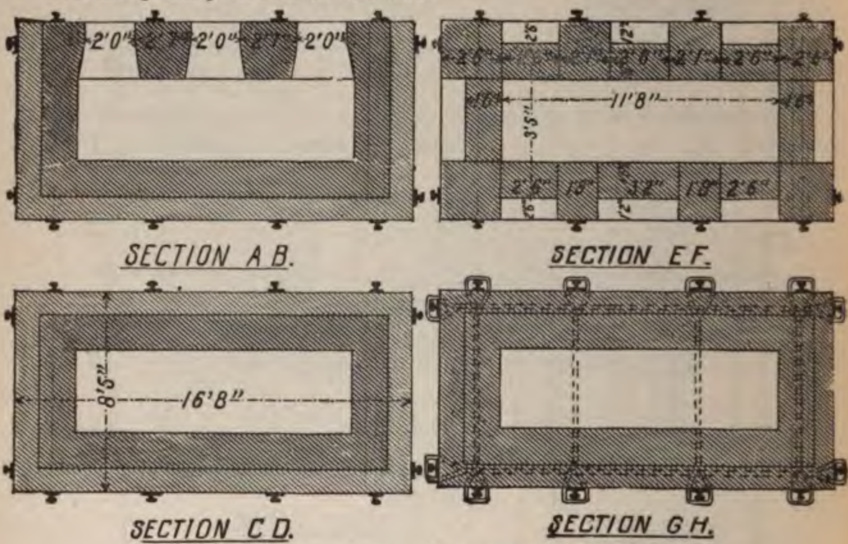


Fig. 48.—The Orford "Raschette" Furnace.

The chimney should never be made smaller than here shown, and if a vertical down-take is used, connected with flues for the saving of the flue-dust, its dimensions should be increased one-third. The latter construction is much preferable to the simple vertical chimney, and is absolutely essential where anything but the poorest material is smelted, as the loss in flue-dust, owing to the enormous volume of blast peculiar to this practice, is very great—especially as a large proportion of the charge often consists of fine ore, it having been found that these large rectangular furnaces are peculiarly adapted to the treatment of that material.

The tuyeres consist of rather heavy, galvanized sheet-iron—No. 18—and are connected with the vertical branches of the main blast-pipe surrounding the furnace, with thick duck tuyere-bags, soaked in a strong solution of alum to render them less inflammable and to fill the pores of the cloth. Their diameter may vary with the character of the ore under treatment, but is usually from five to six inches, the pipes being merely thrust a short distance into the square orifices left in the brick-work, and made tight with plastic clay.

There remains nothing in the construction of this furnace that cannot be plainly seen from the illustrations, and the discussion of

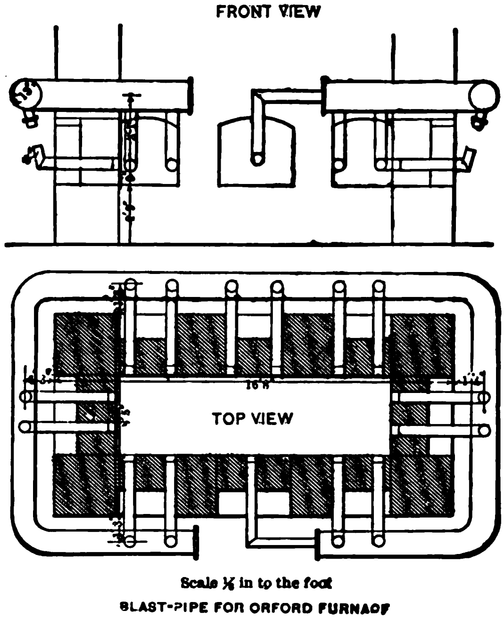


FIG. 49.

its management from the time when taken in hand by the smelter will now be proceeded with.

It is frequently customary to form the bottom of a solid mass of fire-brick, placed on end, and brought up to within 10 inches of the tuyere openings, sloping slightly toward the slag-run in the center of the front wall.*

* The practice of basing the bottom upon an arch built over an open space below must be strongly condemned, as it will simply result in the cutting through of the arch, and the total disappearance of all metal until the cavity is filled, making eventually a solid, but somewhat expensive bottom.

The author has found the following method, practised originally by the Orford Company, far superior to any other, especially where low-grade matte is to be produced, the most difficult of all copper-bearing materials to confine within brick walls.

After filling in the foundation with béton to a foot below the ground level, the furnace bottom is begun by laying two courses of fire-brick on end, and with the closest possible joints. This still leaves a space of from 24 inches to 30 inches to bring the bottom to the proper height, which is filled in as follows:

The furnace and foundation being thoroughly dried by at least four days' brisk firing with brands and similar material, enough coke is dumped into the red-hot shaft to fill it to a point some three feet above certain temporary openings that should be left in the brick-work while building. These openings correspond in size, number, and position with the permanent tuyere openings, except that they are some 8 inches lower and directly beneath the regular orifices, which, for the present, are plugged with clay.

Some six or eight tons of calcined quartz crushed to the size of chestnuts and mixed with about 5 per cent. of fusible slag, are spread upon the coke; and as soon as the latter is properly on fire above the temporary tuyere openings, the blast-pipes are put in place, and a light blast is continued until the coke is burned away, and the sticky, half-melted charge threatens to flow into the tuyere openings. The unconsumed coke and excess of quartz are removed through the breast panel—which was built up temporarily of 4-inch brick-work; and the furnace, being tightly closed, is allowed to cool very gradually for twenty-four hours or more.

If the operation is successful, the bottom will be as solid and infusible as can be made, nor will any attempt at the substitution of basic material for quartz, in consideration of the probably highly ferruginous character of the slag to be produced, result in any improvement on the plan recommended.

It is probably as good a bottom as can be made, although, as will be later seen, it offers but little resistance to a hot low-grade matte, when produced at the rate of from 30 to 50 tons daily.

The furnace being thoroughly dried and heated, blowing in may follow at once, it being only necessary to plug the temporary tuyere orifices, fill the shaft with coke to a point some 3 feet above the permanent tuyeres, and allow the fire to ascend to these openings before filling the shaft with alternate layers of charge and fuel, and putting on a light blast. All this may be done the night before starting, and the forehearth, with siphon-tap, must then be arranged. (See section on "Forehearths.")

The full burden may be reached after feeding two quarter charges, four half charges, and eight three-quarter charges, slag being substituted for ore to a considerable extent, until the condition of the furnace warrants the employment of the normal mixture.

This is shown by the gradual change of the color of the slag from a dull red to a yellowish white; the entire ceasing or great diminution of smoke arising from the slag; a certain peculiar viscosity (except in very basic slags) when it falls into the pot; a general brightening of the tuyeres, succeeded by the formation of short noses, perforated abundantly with bright holes; and a steady and rapid sinking of the charge.

Although the charging of the blast-furnace is always one of the most important manipulations belonging to this apparatus, it is doubly the case with the furnaces now under discussion.

While the walls of the water-jacket are thoroughly protected and entirely unassailable, the mason-work of the brick furnace is completely exposed, and any error in the proportion of fuel to ore, or in the manner of charging, is sure to be followed by serious results.

This is, strange as it may seem, peculiarly the case with a siliceous charge, and nothing can more clearly illustrate the proper method of working than a brief description of an irregularity that is constantly liable to occur, and that will be quickly recogrized by all practical cupola smelters.

An imaginary case will be assumed where a newly blown-in furnace in good condition, but with a slightly too siliceous charge, begins to become too hot in one end, through some slight irregularity of feeding, or through an improper proportion of ore to fuel —either too much or too little of the same producing very similar effects.

The attention of the foreman will be called to the fact that one of the end panels is becoming very hot, which, as it consists of 18 inches of fire-brick, shows either that the inner temperature is much too high or that the bricks have already been thinned by burning.

A glance into the tuyere opening shows that a heavy black nose has already formed, resulting from the fusion of the fire-brick above, which form a crust almost impervious to a steel bar, and exceedingly infusible.

A consultation with the man who feeds that end of the furnace will elicit the information that that portion of the charge is sinking very slowly, and that the heat is rising to the surface.

At the same time, the blast-gauge will show an increased tension, owing to the blocking up of the tuyeres that supply that portion of the apparatus, and the agglomeration of the charge above, owing to the rapidly ascending temperature.

The already too siliceous slag is rendered still more infusible by the admixture of silicate of alumina from the melting fire-brick; and the high temperature and powerful reducing atmosphere, resulting from the almost stationary condition of this portion of the charge, soon begin to reduce metallic iron out of the slag, and even from the matte, the sulphur being driven away to a considerable extent by the powerful blast, high temperature, and slow removai of the molten products.

The slimy, half-fused metallic iron is soon recognized by the bar which is constantly thrust into the choked tuyeres, and the inexperienced metallurgist, following the teaching of all our best text-books, reasons that the reduction of iron comes from too highly ferruginous a charge, and destroys all hope of improvement by cutting off a portion of the iron from the charge fed into that end of the furnace.

This further diminution of the oxide of iron, and consequent necessary increase of temperature to melt the more and more infusible slag, soon bring about the exact conditions prevailing in an iron-ore blast-furnace. Metallic iron is reduced in large quantities, while the temperature is raised several hundred degrees, before the slag—now virtually an acid silicate of alumina and lime—will become sufficiently softened to run at all. In the meantime, the furnace wall, at the panel, is burned nearly through; jets of blue flame appear at every joint and crevice, and the most superficial examination shows that the process is extending into one or the other of the corner columns, threatening the stability of the structure, and still more alarming the person in charge. The column of ore in that end of the furnace hardly sinks at all; the heat is ascending to the surface of the charge; and the general increased stickiness of the rapidly lessening slag-stream, increase in tenor of the matte, and deposition of lumps of metallic iron in one or both compartments of the forehearth, show that the end is not far off and unfold the near prospect of a chilled furnace, and the probable presence of a block of half-molten ore and iron that is almost impervious to tools, and may result in the entire abandonment and destruction of the furnace.

This is one of the most common and well-known occurrences in small furnaces and with inexperienced metallurgists, and might just as well happen to the large furnaces now under discussion, were it not, fortunately, that their construction and management are not likely to be undertaken except by men of experience, and also that, owing to their greater size, a threatening—or even established—chill is much more easily managed than in the case of the smaller cupolas, whose contracted shaft is filled up solid almost before one is aware that anything is going wrong.

Owing to the great area of the Orford furnace, a considerable portion of the shaft may be completely blocked by a chill, while a brisk fusion is progressing in the other half, giving an opportunity, by the use of skill and experience, to gradually smelt away the solidified portion and eventually bring matters back to their normal condition.

Returning to the imaginary case that has just been followed to a disastrous termination, the writer will endeavor to show how such a catastrophe may be averted, and will describe the course of events as they have occurred scores of times to every practical smelter.

The moment that it is noticed that one end or corner of the furnace is becoming abnormally hot, and that the column of ore corresponding thereto is sinking slowly, the tuyeres belonging to that portion of the shaft—from one to three in number—are immediately removed, and the openings slightly plugged with clay. At the same time, several charges of the most fusible slag—that from matte concentration and containing a very high percentage of iron is best—are given, in place of ore, and the whole furnace is most carefully watched, to learn whether the burning is due merely to some local irregularity in feeding, or whether some important point affecting the whole process is at fault; such as too much or too little fuel in proportion to ore; improper composition of slag; incorrect feeding; too strong or too weak a blast, etc., etc.

Experience alone can qualify the metallurgist to quickly and correctly detect the cause of the trouble and apply the appropriate remedy; but in any case, if, after taking the precautions enumerated and waiting a sufficient time to get their full effect, the burning still continues, it becomes evident that the trouble is deep-seated and of some extent.

Vigorous measures are therefore required to stop the melting of the brick-work above the tuyeres, and not only to cool down the heated end of the furnace, but also to repair, as far as possible, the damage already done to the panels—or even to the corners of the main columns.

Still keeping the offending tuyeres closed as already described, a full charge of siliceous ore should be fed in such a way that it will sink to the indicated spot. This may be given either with or without coke, or may be followed by a second or third, or even a greater amount, as the crcumstances indicate; proceeding with extreme caution, and allowing some two hours to intervene between charges.

The author has found it necessary to charge as much as 11 tons of almost pure silica—quartz with specks and veinlets of carbonates and oxides of copper—into one corner of an overheated furnace, and this entirely without coke, before the gradual cooling of the external walls, normal and even sinking of the charge, and lowering of the temperature at the charging-door, indicated that the mischief had ceased.

The office of this siliceous addition is not to render the slag in general more siliceous. This would only bring about the evils already indicated, and probably cause a heavy reduction of metallic iron. Its object is rather to produce, by the sudden arrival of such a body of cold, infusible material, such an overwhelming effect as completely to cool down that portion of the shaft, the silica itself softening somewhat and remaining for the most part in the corner of the furnace corresponding to the point over which it was charged. It attaches itself to the walls and bottom, and fills up the cavity caused by the fusion of the fire-brick, lowering the temperature at the same time to a considerable extent, but producing no marked effect on the general character of the slag.

When this operation is successful, as is usually the case, the thinned and heated brick-work is virtually restored, the deeply excavated bottom is filled up to the general level, and matters resume their normal condition, all irregular bunches and protuberances of the siliceous addition that may have adhered to the furnace walls becoming gradually melted away and smoothed down, until the interior mason-work, if visible, would be seen to have almost assumed its original appearance.

Such a result may seem very doubtful, and, in fact, the whole operation may appear to partake too much of the marvelous to those unfamiliar with such practice. The author would hesitate before describing the foregoing operation as a matter of general everyday occurrence, were it not that it can be vouched for in its entirety by a considerable number of well-known and reliable gentlemen. This practice, as initiated by certain members of the Orford Company, already mentioned, has spread until it is now a well-known and recognized part of our local copper metallurgy. The skill attained by certain foremen in managing these very large furnaces is quite remarkable, and far beyond anything described in this treatise.

While the imaginary case just described in detail represents only one of the various accidents peculiar to all forms of blast-furnace, it still is at the bottom of a very large proportion of the instances of "freezing," "choking-up," "burning-out," etc., etc. Paradoxical as it may appear, the two common accidents of "burning-out" and "freezing-up" are closely connected, and in reality only two different stages of the same morbid process. The young metallurgist cannot overestimate the importance of the fact that it is quartz in one or another of its forms, in a furnace that is not intended for a siliceous charge, that is the most frequent cause of smelting difficulties and disasters. Seven out of the last eight cases of metallurgical difficulties for which the writer was called upon to prescribe, were due to this cause.

In spite of the frequency and apparent simplicity of this difficulty, some smelters of experience never seem to have learned the cause, and attribute the slow and irregular running of the cupola and the frequent filling up of the crucible with sows to "too much iron in the charge"—"too much sulphur"—"magnesia in the limestone flux," etc., when in almost every instance a mere ocular examination of the slag is sufficient to show that silica is at the bottom of the trouble. No apology is needed for emphasizing this point when men considered as expert metallurgists are constantly falling into this error.

It is especially during such accidents and irregularities that the great advantages of these very large furnaces become fully apparent. Where a small shaft would soon be completely and irretrievably choked, necessitating the great expense of blowing down and subsequently chiseling out the half-fused mass of ore and cinder, no large furnace, in any instance known to the author, has ever become so blocked up and filled with a chill that it has not been quite easy to save it by using appropriate means. Even though one end be completely blocked, there is always ample space at some points of its eleven-foot shaft to permit the descent of the charge and retain a sufficient number of tuyeres intact to gradually melt out the chill and restore the shaft to something like its former dimensions. Some considerable irregularity of form naturally results from repeated manipulations of this kind; but so long as sufficient area remains at the tuyere level, and no projecting masses impede the regular descent of the charge, no diminution of capacity need follow, nor increase of difficulty in managing the furnace.

The accompanying sketch gives a tolerably correct view of the shape of one of these large brick furnaces at the tuyeres upon its blowing-out for repairs after a continuous campaign of 8½ months, during which time over [illegible] tons of exceedingly ferruginous ore were smelted in it, yielding a very low-grade matte and slag aver-

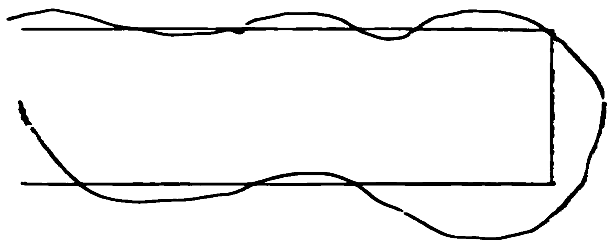


FIG. [illegible].—THE RECTANGLE SHOWS THE SHAPE BEFORE THE CAMPAIGN; THE IRREGULAR LINE, AFTER THE CAMPAIGN.

aging about [illegible] per cent. silica and over [illegible] per cent. protoxide of iron. As it is drawn to a scale, the extent of the irregularity is easily appreciable, the original dimensions being 3 feet 3 inches by 11 feet 4 inches.

In fact, the full capacity of this type of furnace, when smelting a basic ore, is not reached until the walls are burned out to a considerable extent, which may indicate the policy of widening the furnace in the first place. When smelting a siliceous ore, or when a large proportion of fines is present, the gain in width is accompanied with a decrease of temperature and irregularities in the descent of the charge—circumstances that soon rectify the trouble by adhering to the walls and filling up the shaft again with a rapidity that may be disastrous if not observed and remedied in time.

As has been already briefly mentioned, the cutting down of the bottom and piercing of the foundation-walls is an accident that sometimes occurs, although usually only when the charge consists of a very fusible unroasted ore, producing a matte of low grade—from 25 per cent. downward—whose fiery and corrosive qualities are well-known to all furnace-men. It is to the great quantity, as well as corrosive quality, of this substance, and this usually in connection with a basic slag, that this destructive process is due; and in spite of much care and expense bestowed on the matter, no material has yet been found that will withstand a daily production of from 20 to 45 tons of this intractable product. But a means of lessening its destructive action, as well as of greatly prolonging the life of the entire structure and rendering its management much easier, has been discovered and quite generally adopted, being first brought into notice by Mr. John Thomson, of the Orford Company. It consists in duplicating the furnace plant and running each individual cupola only ten or twelve hours of the twenty-four. This is a scheme that seldom recommends itself to one on first hearing, but, after a thorough trial, will be found to possess numerous important advantages, while its only drawback is the increased first cost of the plant—a trifling consideration in comparison with the large interests usually at stake.

A mere doubling of the cupola plant is sufficient to overcome the difficulties mentioned; but if it be desired to reap the full advantages of the scheme, a corresponding increase should be made in the blast apparatus. This being effected, the entire smelting process may be confined to the daytime, avoiding the difficulties and drawbacks of night work, saving the wages of one or more foremen, and rendering it possible for the manager to retain that complete personal oversight of the smelting process that is unattainable when half of it is concealed from his inspection. If this were the only benefit derived from the above plan, it would in most cases be well worthy of adoption; but the advantages accruing to the furnaces themselves, as well as to the entire process, are too numerous and far-reaching to be thoroughly explained in this treatise.

In the first place, the cutting down of the furnace bottom is usually completely remedied by the long and ever-recurring periods of complete repose, during which the thinned brick-work is again sealed by the chilling of the molten products; the hearth is renewed by the solidification of the matte and slag still remaining in the cavities of the hearth; the overheated brick-work cools from the outside to such an extent that the area that to-day has given constant annoyance by its obstinate burning, with the constant threat of finally breaking through and causing serious trouble, will to-morrow be found as cool as, or cooler than, any other portion, owing to the thinness of its walls; and various slight difficulties that are pretty sure to occur in the course of a long run are averted before they become of importance, while the trouble begins at a new point, only to be again averted before it has gained serious headway. This is by no means an uncommon or imaginary case, but a matter of frequent occurrence, and these lines are written after several years' trial of both the constant and intermittent method of smelting, the experience of others who have fairly tried this plan, in connection with large brick furnaces, being equally favorable.

The writer's attention was first called to this matter in 1871, when noticing the almost invariable improvement in behavior and capacity that succeeded any accidental stoppage of cupola-furnaces that he was then managing. The ores were exceedingly bad and siliceous, and the difficulties detailed in the preceding pages followed each other with disheartening regularity and frequency. Great pains were taken to secure a steady and uninterrupted run, fears being entertained that any stoppage would be disastrous to the furnace in the more or less critical condition that seemed to be its normal state; but after finding that the benefits following any temporary stoppage of the machinery had become so obvious that the foreman was in the habit of purposely causing slight accidents in order to help his furnace out of some particularly critical situation, it was decided to adopt the practice of stopping for two or three hours whenever the ordinary incidents of burning out, etc., became unusually critical. This habit was carried further and further, proceeding with caution and gradually lengthening the stoppages, until it came to be considered an almost universal remedy, and was as often applied for chilling or freezing up as for the opposite condition of affairs, and no misfortune ever arose from its reasonable application.

This practice, like every other, must be used with care and judgment, and may easily be carried to an extreme, but, as a rule, is the least dangerous measure that can be adopted with a badly acting furnace of large area. A small furnace might easily chill in a few hours, so that the length of the period of repose must be proportioned to the size of the shaft and to the cubic contents of the heated material. The thickness of the walls must also be considered, as the rapidity of the escape of heat depends upon the thickness of the brick-work. It is hardly necessary to say that every orifice and crevice about the furnace must be tightly sealed, the tuyeres being removed, and their openings, as well as the slag-run, being tightly filled with damp clay, while the brick-work in their vicinity must be searched for possible cracks, and all such openings carefully plastered over. Otherwise, the incoming currents of air would gradually burn away all the fuel contained in the charge, leaving the furnace in a hopeless condition. If it is to stand still any length of time, such as over night, a little extra coke should be given an hour or two before stopping, so that there may be an abundance of fuel in the bottom of the furnace. A small charge of basic slag should also be given; and as soon as the blast is taken off, the basin or forehearth tapped, and all openings sealed, the surface of the charge should be covered with a layer of fine coke, over which is spread an inch or two of fine, fusible ore. The slag-hole connecting the furnace with the forehearth should be thoroughly cleared out; the layers of chilled slag and ashes, by which the blowing through of the blast is prevented, removed, and the channel itself filled with fine charcoal or coke, well rammed in with a "stopping pole." This is rendered impervious to air by an exterior plug of clay, and the forehearth, while still hot, being scraped clean of all half-fused masses of slag or reduced iron, and everything being prepared for the morrow's work, the cupola may be left in charge of an experienced watchman—preferably an old smelter. On the ensuing morning, a light blast is put on, and the channel being cleared out, slag will flow in from five to ten minutes, while in half an hour the furnace will be in normal condition, and in most cases smelting more rapidly and satisfactorily than when left the previous evening.

The extreme length of time that a large furnace may stand in this way without injury is unknown to the author. Much depends on the fusibility of the charge, the character of the fuel, the more or less perfect exclusion of all air, and probably also upon the quality and amount of sulphide compounds present, whose gradual oxidation may sustain the vitality of the charge for a much greater length of time than if absent. The following instances, from personal experience, show that a considerable delay is permissible.

A furnace running on a fusible charge of calcined pyritic ore was shut down Friday noon, on account of an accident to the engine. Further examination showed the accident to be of such a nature as to cause a delay until the succeeding Wednesday night —5¼ days—at the end of which time a light blast was applied without much hope of a favorable result, although the coke on top of the charge was hot and glowing.

There seemed a good deal of obstruction to the blast at first; but in twenty minutes, a cold, thick slag began to run, which gradually improved, until the furnace resumed its normal condition and capacity in about eight hours. The charge had sunk about two feet in the furnace during this period of repose. The grade of the first tap of matte (the siphon-tap being impracticable in this condition of affairs) was 46 per cent., the ordinary average being from 28 to 29 per cent. The succeeding tappings gradually decreased—going successively 42, 37, and 34 per cent., the normal grade being reached soon after the furnace had regained its usual capacity.

Periods of 4 days, 3¾, 3½, 3, and of less time, appear in the writer's notes, the only serious accident, occurring during one of the shorter periods, being caused by the falling out of two of the tuyere-plugs, whereby a current of air entered the furnace for twelve hours before being discovered. The coke was completely burned out of the lower portion of the charge for about two-thirds of that part of the shaft nearest the opening; but the furnace was eventually saved by blowing lightly into three tuyeres at the opposite end, which were still supplied with fuel, and little by little smelting out the entire half-fused block of charge. Much benefit was derived by introducing coke into the furnace through such tuyeres as seemed to warrant the trouble. Owing to the great size of the tuyere openings (6 inches), this was easily effected, and the smelting much facilitated. In fact, if any cavity in the semi-fused mass could have been found at any point accessible to the blast, nothing would have been simpler than to break a hole through one of the brick panels and fill the opening with coke. The author has done this in later instances with very satisfactory results, a cavity opposite the tuyeres having been formed by dragging out a lot of the stock, from which the coke had burned so gradually as not to fuse it.

Space is wanting for a description of the use of petroleum, gas, and other concentrated fuels for similar purposes, as the writer's own experience with such measures has been entirely unsatisfactory, nor can he find any record of successful cases in the annals of American copper smelting.

The most herculean efforts are warrantable when any reasonable probability exists of the saving of an iron furnace from complete chilling up; but in copper smelting, the comparative cheapness and simplicity of the structure itself, and the certainty of being able to remove the worst chill by mechanical means in a comparatively short time, render such unusual and expensive measures less important.

The oxidation of the sulphides in the charge during the period of repose is an element of some importance, although seldom so striking as in the case just mentioned. Still, the closing down of the cupola over night is invariably accompanied with a perceptible rise in the grade of the matte produced during a certain period succeeding; being greatest at first, and gradually diminishing as the contents of the furnace are replaced with fresh ore. This increase in richness is at first seldom less than 5 per cent., diminishing rapidly, however, as the ore nearest the bottom of the charge has experienced the most thorough oxidation.

Though apparently a trivial matter, this enrichment of the matte is a direct pecuniary gain, and, according to a rough estimate, will offset the interest on the capital necessary for the double plant several times over in the course of a year.

Another useful and frequently applied remedy for various irregularities in cupola smelting is the so-called "running down" of the furnace, by which is meant a mere cessation of charging until the column of ore and fuel has sunk to a point far below its normal limits. The shaft is then rapidly filled with the usual alternate charges of ore and fuel, and everything goes on as before.

This practice is sometimes of great advantage, obstinate irregularities often being conquered thereby, and the normal condition of things resumed. It is especially useful when it is desired to create a sudden and profound lowering of temperature at some point where a serious localized burning is taking place; for the exposure of the naked inclosing walls of the shaft renders it possible to deposit the batch of ore that is used to cool the walls in the exact spot where it is needed; and it is possible to use for this purpose, under such circumstances, an easily fusible ore or slag, instead of the highly siliceous material that is usually selected when this process of cooling down is undertaken blindly from above.

Wall accretions may also be reached in this manner, the charge being allowed to settle until they are exposed, whereupon they may be removed by a long,bent steel bar introduced through one of the charging-doors, the glowing interior being cooled down, if necessary, by sprinkling with water.

Still another means of remedying the cutting-down of the furnace bottom has been mentioned in a former section, but is sometimes useful in connection with the large brick furnace. This is the introduction, through the tuyere openings, of ore or sand, which, being both cold and the latter infusible, will not combine with the slag, as it is already below the smelting zone; but will simply remain in place and assist in building up a new bottom. By this means, even the molten masses present may be partially solidified and a great advantage gained in a short time. The author has occasionally tried the introduction of water in the same manner and for the same purpose, taking as a guide the very decided local chilling produced by a leaky water-jacket; but the results, though locally satisfactory, are not sufficiently extended, while the operation itself, especially in connection with a low-grade copper matte, cannot be recommended to any who object to certain and frequent explosions of considerable force.

In connection with the measures already detailed for keeping the furnace in proper condition, may be mentioned the external repairs that it is feasible to execute while the furnace is still in blast. Not all smelters are aware of the very extensive repairs that may be carried out without stopping the blast more than a few hours; the length of the campaign often being doubled by the construction of a new panel, the repairing of a pillar, and other familiar and inexpensive operations. These are of too extensive and varied a nature to be enumerated in detail; but a few of the teachings of experience will throw some light on the practice in general.

The replacement of one or more panels that have become so thin as to threaten a constant breaking through of the charge is a simple, though very hot and laborious task.

All needful material for the renewal being prepared and collected on one spot, the blast is shut off, the forehearth tapped, and the condemned brick-work at once broken in with sledge and bar. So much of the glowing charge as is necessary is at once dragged out of the opening with long hoes and rakes, and sprinkled with water so that the men can stand on it to work.

When the bricks have been removed to the extent deemed necessary, the cavity left in the column of stock is quickly filled with dampened coke, a few wooden slats being wedged across the opening, to keep the fuel from falling out.

The most important measure is to obtain a solid foundation for the new wall, and to accomplish this, all accretions of slag and metal, of which the old wall largely consisted, must be chiseled away until sound brick-work is reached, which being leveled with thick fire-clay, offers a proper starting-point. The work must proceed with great rapidity, as the passage of air through the opening will soon consume the fuel in the charge. Little attention is paid to neatness, or even regularity, so long as strength and tightness are obtained. If the work promises to occupy more than two or three hours, the opening should be closed at the beginning by a thin plate of sheet-iron tightly cemented at the edges with clay, outside of which the new wall is raised. When all is completed, the sheet-iron—unless already consumed—is cut away opposite the tuyere openings, and the blast is put on at once, there being no necessity of waiting for the work to dry, as the heat from the furnace will evaporate all moisture quite as soon as is desirable.

By this means, extensive repairs may be executed on any portion of the furnace, it being even possible to put in a new bottom, or repair the foundation walls, by suspending the charge on bars driven transversely through the furnace. When possible, the ashes of the rapidly consumed fuel should be cleared out before starting again; but there are but few instances where it will not be found better to blow out the furnace when such radical repairs are required.

The final blowing out of the large furnace presents no peculiar features. The blast should be lessened as the charge sinks, and as soon as slag stops running, the breast-wall, and, if expensive repairs are imminent, some of the rear and end panels should be knocked in, and all stock and fuel dragged out, until a tolerably even bottom is reached, which needs no preparation for the succeeding campaign.

Any burning out of the brick pillars that form the main support of this furnace should be carefully watched and repaired before it has proceeded to a dangerous extent. This burning is sometimes so obstinate that when it is important not to stop the furnace or blow out, it is necessary to support the superincumbent brick-work with props and braces, which should remain in place until the pillars have been restored to their former strength.

Estimates of the cost of building one of these large brick furnaces of the Orford type will be found in this chapter.

There remains to be still considered the application of water tuyeres and other cooling devices to furnaces constructed of brick or stone.

The author's own experience is entirely in favor of the employment of properly constructed iron, or better, bronze or copper tuyeres, containing a space for the introduction of water. In Colorado and other places, he has used water tuyeres with invariable satisfaction, the only drawback being the frequent cracking of the cast-iron, which is now overcome.

While they offer little or no protection to the furnace wall, they are indestructible themselves, and by delivering the wind at a fixed point, even though the walls may be eaten away all about them to the depth of a foot or more, they remove the point of greatest heat from the wall itself, and practically retain the smelting area at the same invariable size, the latter being practically bounded by vertical planes passing through the nozzles of the tuyeres.

It is also possible, if desirable, to project them into the interior of the furnace to a distance of several inches from the walls. Although this practically diminishes the size of the smelting area, it saves the walls from burning, and in case of a weak blast or of an unusually dense charge arising from a large proportion of fine ore, may render practicable the smelting of material that would be impossible under other circumstances.

They were tried on the first large Orford furnaces, but failed, owing to the severity of the winter and other accidental causes, rather than from any fault due to the tuyeres themselves. Their construction and management are too familiar to require further explanation in these pages.

The surface cooling of the brick-work by means of a spray of water on the outside has been tried on many occasions and with various forms of apparatus. It has rarely given satisfaction, and, in the writer's opinion, is as dangerous and worthless a device as can well be imagined. To those familiar with the results of contact between water and molten matte, it is not necessary to bring up any further arguments to condemn a device that can only be accompanied by a constant wetting of everything in the vicinity of the furnace.

Besides, the idea itself is an extremely faulty one, as, owing to the non-conductivity of fire-brick, a wall less than a foot thick may continue melting on one side, while its other surface is constantly sprayed with cold water.

All devices of this kind, in which the water comes in contact with the free exterior sufrace of the furnace wall, are, in the author's opinion, worse than useless, and likely to be accompanied by most dangerous results.

ESTIMATE OF COST OF LARGE BRICK BLAST-FURNACE.

Excavation for foundation: 1,000 cubic feet at 8 cents...		$80.00
Foundation of béton		65.00
	Cubic feet.	
Total fire-brick for furnace proper	1,640	
Lining for cross-flue and down-take	540	
Forehearth, etc.	45	
Total	2,225	
At 18 brick per cubic foot = 40,050 at $40 a thousand		1,602.00
Red brick for down-take and flue: 16,800 at $8		134.40
6½ tons fire-clay at $8		52.00
6 casks lime at $1.50		9.00
2 tons sand at $1.50		3.00
Old rails for binders: 180 yards at 80 pounds a yard = 14,400 pounds at ¾ cent		108.00
Tie rods for furnace, flue, and down-take: 620 feet of 1¼ iron = 2,480 pounds	Pounds. 2,480	
Loops, nuts, etc.	166	
Angle iron for down-take	172	
Wrought-iron rods, etc., about forehearth	66	
Total	2,884	
At 2 cents a pound		57.68
Castings:	Pounds.	
3 feed-door frames	792	
Damper and frame	455	
Plates for forehearth	560	
Slag and matte-spouts	80	
Plates for charging-floor	1,260	
Miscellaneous	420	
Total	3,567	
Brought forward		2,111.08

Carried forward	$2,111,08
At 2½ cents a pound	89.17
Material and labor for arch patterns and other carpenter work	32.40
Labor:	
Mason, 88 days at $4	352.00
Ordinary labor, 102 days at $1.50	153.00
9½ days, smith and helper	47.50
Blast-pipe and tuyeres	136.00
Cloth for tuyere bags and labor	3.80
Superintendence	120.00
Miscellaneous	65.00
Grand total	$3,109.95
Tools essential to furnace, steel, and iron bars, shovels, rakes, hammers	55.90
15 slag-pots at $13.50	202.50
4 iron barrows at $9.00	36.00
Manometer	2.50
Total	$296.90

The above estimate is exclusive of main blast-pipe, blower, motive power, hoist, and chimney or dust-chambers; the allowance for cross-flue and down-take being sufficient to cover cost of chimney in those exceptional cases where no provision is made for catching the immense amount of flue-dust generated in this method of smelting.

A compact and economical hoist and ample provision for a large charging-floor and generous bin room are essential to convenient and economical work.

CHAPTER XIII.

GENERAL REMARKS ON BLAST-FURNACE SMELTING.

THE capacity of a blast-furnace is dependent upon many varying causes, and is to a considerable extent independent of shape or size, though its tuyere area is, of course, the most important factor in determining the amount of material that can be passed through it.

Next to the fusibility of the charge, the pressure and volume of the blast have the principal influence in determining this point, assuming always that the fuel used is of sufficient strength and density to permit the full pressure of wind that may be found most advantageous.

Nothing can be more striking than the change in the rate of smelting of a large cupola-furnace as the wind pressure is diminished or increased.

The author has taken occasion during the smelting of a fusible charge, and with the furnace in perfect condition, to ascertain the difference of capacity effected by changes in the strength of the blast.

As the influence of the change is almost instantaneous, it is easy to arrive at such figures with considerable accuracy, measuring the capacity by noting the number of pots of slag produced during periods of an hour each, and with varying wind pressure.

The following table shows the result of these experiments in a compact form, repeated sufficiently often under varying conditions to establish their comparative accuracy.

It should be mentioned that, in order to insure the accuracy of each observation independently of the condition of the furnace previous to the experiment, which might have been influenced by the preceding test, nearly all the trials were made at different times, but with the furnace as nearly at its normal state as possible, and running under its ordinary pressure of blast—about 10 ounces per square inch:

No. of Test.	Blast Pressure in Oz. Per Sq. In.	Production in Tons. Per 24 Hours.	Assay of Slag in Copper.	Condition of Furnace at Close of Experiment.
1....	½	16¼	0.27	Very hot. All tuyeres bright.
2....	1	21	0.35	Very hot. All tuyeres bright.
3....	2	31½	0.30	Very hot. All tuyeres bright.
4....	3	44	0.31	Slag hot and smoking. Tuyeres bright.
5....	4	64	0.31	Slag hot and smoking. Tuyeres bright.
6....	6	86¼	0.51	Slag hot and smoking. Tuyeres bright.
7....	8	87½	0.40	Slag still hot, but not quite so strikingly
8....	9	91	0.42	so as with lower pressure. Tuyeres sat-
*9....	10	99½	0.42	isfactory, but beginning to form noses.
10....	12	113	....	
11....	13	111	....	Less hot. Decided noses.
12....	14	116	0.66	Much cooler. All tuyeres require opening.

* Normal pressure and slag assay.

These tests, although not entirely uniform in every respect, are still quite regular and agree closely with many previous observations.

With the highest available blast, 14 ounces per square inch, the production still increases, though only slightly above the normal capacity, but it is evident more wind is introduced than can be consumed by the fuel; a lowering of temperature occurs, as distinctly shown by the appearance of the slag; and thick, hard noses are formed about each wind stream, which would soon obstruct the blast, and probably cause a general chilling of the furnace.

Judging from this series of tests, as well as from numerous former trials, when smelting both lead and copper ores of many different varieties in cupolas of various sizes and under very varying conditions, it seems advisable to limit the blast pressure to the point just indicated except where furnaces are to be used simply for melting, regardless of any possible oxidizing effect. In no single instance has anything more than a temporary increase of capacity accompanied a blast pressure above 12 ounces per square inch, and the rapid cooling of the furnace and formation of heavy and solid noses have soon brought the experiment to a termination.

It seems, therefore, that a pressure of from 8 to 12 ounces, with a tuyere diameter of from 3 to 5½ inches, is best suited to the ordinary conditions of copper smelting.*

The employment of soft-wood charcoal or other fragile fuel may make it necessary to diminish even this light pressure, while anthracite may demand a more powerful blast for its most economical use.

* These words were written for the earlier editions of this book, and since that time experience has taught the important effect produced by blast pressure upon the oxidizing influence of the blast-furnace.

I have many times used wood in two-foot lengths to replace a portion of the coke in the blast-furnace, though merely to tide over a time when coke was scarce. I have invariably noticed a decided rise in the grade of the matte when smelting with wood. Mr. Herbert Lang* gives some interesting information on the subject, as follows:

"Cordwood, sawn in blocks of a foot in length, is a regular constituent of our fuel charge at Mineral, Idaho, our work being the matting of silver ores by fusion in a blast-furnace. The furnace is a round water-jacket furnace, of 36 inches diameter at the tuyeres, and the charge of smelting mixture weighs 950 pounds, requiring 110 pounds of Connellsville coke to drive it. I replace half of this coke with 135 pounds of firwood, cut from dead and apparently perfectly dry trees. This mixture produces as high a smelting temperature as all coke, whence I infer that the smelting effect of a given weight of wood is to that of the same weight of coke as 11 to 27, or 1 to $2\frac{5}{11}$. A cord of wood sawed ready for use weighs 2,340 pounds, costs $5, and is equivalent to 866 pounds of Connellsville coke, which, at $25 per ton, costs $10.92, or rather more than twice as much as wood per unit of smelting power. The saving by the use of wood plus coke, over coke alone, is therefore 75 cents per ton of ore. The principal advantage, however, is not in the saving of cost, but in the fact that a great deal of sulphur is burned off by the wood, thus allowing the use of a greater proportion of sulphide ores in the charge, which is a point of great moment, as such ores predominate here, and we are as yet unprovided with roasting apparatus. To offset these advantages, the wood produces a great deal more flue-dust—twice as much, I should think—and reduces the smelting capacity about one-third. With the fuel mixture described, I can carry only six ounces of blast; but the furnace keeps in good condition above and below, the tuyeres remain unaffected, the slag is hot and reasonably free from valuable metals, and the conditions of successful smelting are met in all respects, except as to the serious reduction of tonnage.

"Mr. Dwight, in his comments upon Mr. Neill's paper on 'The Use of Stone Coal in Lead Smelting,' appears to infer that the coal has to be converted into coke inside the furnace before it can perform useful work. I presume he would also infer that wood has to become charcoal before it can do its smelting work, but

* *Transactions American Institute Mining Engineers,* Vol. XX., p. 545.

that such an inference is erroneous appears from the fact that our firwood produces but about 20 per cent. of charcoal, and that of a very poor, fragile sort. Accordingly, 135 pounds of wood would produce only 27 pounds of charcoal, a quantity clearly insufficient to replace 55 pounds of coke. I therefore believe that the volatile constituents of the wood do a considerable amount of useful work in the smelting before escaping from the furnace. The smoke, which is very thick and abundant, has a peculiar nauseating odor, giving no evidence of free sulphurous acid—a circumstance which leads me to believe that the sulphur so largely burned off forms a volatile compound with the organic matters sublimed from the wood, the reaction perhaps furnishing a considerable amount of heat. I presume that the use of denser kinds of wood, such as mountain mahogany, oak, hickory, ash, etc., would give still better results."

Mr. James W. Neill, of Leadville, Colorado, has made some experiments on the use of bituminous non-coking, and semi-coking coal in the lead-silver blast-furnace, that are highly suggestive to copper smelters. I quote from his paper:*

"Bituminous coal has for many years been used for the smelting of iron ores in the blast-furnace. In some districts in Scotland it is used alone, in others it is used mixed with coke. The similar use of certain bituminous coals in the United States has been repeatedly mentioned. In the lead-silver smelting blast-furnace, however, the requirements of iron smelting are not present. Here the general question is, which fuel, or fuel mixture, will permit the most rapid driving of the furnace? The conditions of efficiency in the reduction of iron, of sufficient heat, of capacity to carry the burden, etc., are usually satisfied by any of the commercial cokes of the regions surrounding the lead-silver smelting districts, and it is therefore usually the price which decides the choice of coke. In most of these smelting districts, bituminous coal of non-coking character is very much cheaper than the coke, and its use alone, or with coke, would materially lessen the fuel expense per ton of ore. This saving, if achieved without occasioning other losses in the working of the furnaces, would be net gain to the silver-lead smelter, and the following experience with the use of bituminous fuel is given to show what can be done in this way.

"About 1884, while in charge of the smelting works at Mine

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* *Transactions American Institute Mining Engineers*, Vol. XX., p. 165.

La Motte, Missouri, I ran short of coke, and having a supply of stone coal on hand, I replaced half of the coke in the charge with this coal, continuing its use until a supply of coke arrived. During this period (about twenty-four hours) no noticeable change in the working of the furnace occurred; but as the stone coal was more expensive than the coke, the practice was not continued. With the precedent of various authorities,* and my own brief personal experience, and in view of the circumstance that in Leadville to-day coke costs three times as much as certain kinds of coal, I have recently ventured to experiment with Rocky Mountain coal in the blast-furnaces of the Harrison Reduction Works.

"These furnaces are 78 by 36 inches in size at the tuyere line, have 10 inches bosh in the jackets, and are about 12 feet high from tuyere to charge door. At the time the experiments commenced, the furnaces, having been running some time, were in bad condition from zinc accretions in the upper part of the stacks, and would have to be "blown down" and "barred out" in a few days at furthest; and I reflected that, if the coal should prove impracticable as fuel, this event would only be slightly hastened.

"On January 20, 1891, I replaced on No. 1 furnace 50 pounds of Cardiff coke with 60 pounds of lump coal of a non-coking variety. The fuel charge before the change had been: Coke, 185 pounds; charcoal, 65 pounds. It was now: Coke, 135 pounds; stone coal, 60 pounds, and charcoal 65 pounds. The charge was made on the evening of the 20th. Next morning showed no appreciable change; slag assays were good, but in the afternoon the slag commenced to thicken and get colder, and finally refused to run out of the tap-hole, filled all the tuyeres, and compelled a stoppage of the furnace. On taking out the tap-jacket, I found that the charge had slipped down, filling the basin with raw material, which had stopped the slag. We heaved out a quantity of this raw material and cleaned the tuyeres, and put on the blast again, when the furnace cleared itself without further serious trouble. Much to our surprise, we found the top of the furnace in better condition, the amount of accretions hanging on the sides being much less and the charges settling more evenly.

"On the 22d, as No. 1 continued to do well on this fuel, I put

* T. Sterry Hunt, *Transactions American Institute Mining Engineers*, Vol. II., p. 275; Vol. VII., p. 313; J. S. Alexander, Vol. I., p. 225; A. Eilser, Vol. I., p. 216; Phillip, *Elements of Metallurgy*, p. 250; Groves & Thorpe, *Chemical Technology*, Vol. I.

the same fuel charge upon furnace No. 4, the upper part of which was also in bad condition. Here the result was the same; but by careful watching serious trouble was avoided. Meanwhile, instead of our having to clean No. 1 in a day or two, it ran 13 days after the coal was first put on.

"During the first week in February, all the furnaces were blown down and barred out, and blown in again on the above fuel charge, all starting off nicely. On the 4th of February the charge was changed to: Coke, 120 pounds; stone coal, 70 pounds; charcoal, 70 pounds; and on this charge they ran until the 14th. During this period the coal used was a lump coal from Rouse, Colorado, a semi-coking coal of good quality, a sample giving us 8.91 per cent. of ash. The pecuniary saving was, that in 14 days we replaced 126.98 tons of coke at $8, costing $1,015.84, with 148.15 tons lump coal at $5.50, costing $814.82, a difference of $201.02.

"On February 14th I replaced the lump coal with an equal weight of pea coal from the Sunshine mines. A sample of this gave us 7.72 per cent. of ash; it is a semi-coking coal, and costs us $2.50 per ton. The furnaces ran for the remaining 14 days on the same charge of fuel as during the first half of the month, doing their work nicely. During this time we replaced 126.84 tons of coke at $8, or $1,014.72, with 147.98 tons of pea coal at $2.50, or $369.95, thus saving $644.77. The total saving for the month of February was, therefore, $845.79, which amounts to about 26 cents per ton of material smelted.

"During March we used about the same fuel charge, the coal being part nut, part pea, and part lump, each used separately; and the total saving by the use of these fuels in place of coke was, for March, $1,075.08, or about 30 cents per ton of material. For a few days I increased the amount of stone coal to 100 pounds, using with it 100 pounds of coke and 70 pounds of charcoal; but the furnace did not work well on this charge, the hearth becoming clogged and slag flooding the tuyeres. Whether this was due to the fuel or to bad charging remains to be proved by further tests.

"Aside from the direct saving by the use of the stone coal, we have observed the following advantages in the working of the furnaces:

"*Below.*—The slags appear better reduced and hotter, and the matte separates very well. Slag assays have been, if anything, lower than on the old fuel; and this is particularly the case when the charges are hanging above. The jackets keep hotter and the tuyeres brighter; thus the fuel is more completely consumed; the furnace 'rods' more easily; the crucible keeps open better; the lead is hotter.

"*Above.*—The volume of smoke is somewhat increased and smells decidedly 'tarry,' but does not look different from that of the old fuel; the charges settle much more evenly and the fire does not creep to the top; thus the furnaces seldom flame. As the tops keep much cooler, the production of flue-dust is much smaller, and the losses of metals by volatilization must also be diminished. The furnaces now run about a week longer than formerly before needing to be barred out; and this operation is no more difficult than before; but on blowing them down, the charge now sinks much further before flaming commences.

"The pressure of the blast has not changed materially from that of the old-fuel charge, but now remains more uniform. The necessary reduction of iron can thus always be relied on.

"To the practical furnace-man, these advantages are of almost as much importance as the direct saving in the cost of smelting.

"The short trial with a higher percentage of coal I do not consider conclusive, and I am now working the amount of coal up by gradual steps to see what proportion we can use to the best advantage. The limit will probably be reached sooner on our low furnaces than it would be on higher ones. We are now using 27 per cent. of coal on the fuel-charge, and I believe that on our furnaces the limit will be reached at about 33 per cent., but that on higher stacks the percentage could be carried to at least 50.

"Of course, this will in a measure depend on the quality of the coal used. For instance, I was obliged recently to use coke and coal from Southern Colorado (containing 20 per cent. and 13 per cent. of ash respectively) in place of our regular fuel (containing 11 per cent. and 8 per cent. respectively), and although I made allowance for the difference in fuel value and added flux for the ash, we had much trouble with the furnaces, owing to scaffolding of charges and incomplete combustion before the tuyeres, with consequent bad effects on the slags, etc.

"There are many smelting districts where the difference between the prices of coke and coal is even greater than in Leadville. In such places the direct gain by the use of coal instead of coke will be correspondingly greater. But to practical furnace-men the other advantages above described will be, perhaps, sufficient to induce a trial of this simple experiment, even where the item of saving in fuel cost is not important."

The common claim urged by inventors and manufacturers of smelting-furnaces is, that their apparatus is capable of generating a temperature much higher than ordinary furnaces.

This shows an entirely mistaken notion of the process of smelting, where our constant endeavor is to prevent the temperature rising much above the point necessary for the fusion of the earthy constituents into a liquid and homogeneous slag.

The method of charging is pretty nearly universal, and differs radically from the old practice, where the establishment and preservation of a nose seemed to be the chief aim and end of the smelter's labors.

Both ore and fuel are now pretty generally spread in horizontal layers over the whole area of the furnace, instead of throwing the coke toward the center, while the charge was carefully placed against the walls.

In the brick furnace especially, the position of feeder is one of vital importance, and the experienced furnace foreman will spend a large proportion of his time on the charging platform.

This matter has been discussed and exemplified in the section on large brick furnaces, and is worthy of the most careful study and attention.

The *absolute* size of the charge to be used must vary according to local conditions.

The most important of these are the area and height of furnace; mechanical condition of ore; nature of fuel; and extent of reducing action desired.

Large and high furnaces naturally require heavier charges of ore and fuel; a charge made up almost entirely of coarse material may safely be fed in thicker layers than if composed principally of fine dirt, which opposes a powerful obstacle to the passage of the blast; a heavy, compact coke will bear a much weightier charge than light, fragile fuel, like soft-wood charcoal; and a more thorough mixing of ore and fuel, as effected by using small charges, will undoubtedly bring about a more powerful reducing effect than when the different strata are of sufficient depth to retain their relative position to a considerable depth.

While very numerous exceptions exist, the author prefers, in general, large charges to small ones, having found, as a rule, that the furnace runs more smoothly and regularly, and also that a slight saving in fuel is effected.*

In only one instance has the writer attempted to determine this point by actual experiment; but in the case referred to, the conditions of the trial were particularly favorable for a fair and impartial comparison.

The furnace was a 42-inch water-jacket, smelting a mixture of reverberatory copper slag and fine unroasted pyrites, with gas coke as a fuel. The foreman, who was a most skillful smelter, was directed during the entire experiment to give his attention to the consumption of fuel, using no more than was necessary to attain the best possible results. The change in the size of the charge was made without directing his attention particularly to it. He was thus left to discover any necessity for a change in the weight of fuel.

The experiment was begun with large charges—1,480 pounds of mixture—the relation of the fuel to the same being as 1 to 9.3. This was maintained for 72 hours, the furnace remaining in excellent condition, and averaging 57 tons per 24 hours.

The charge was then reduced to 740 pounds, just one-half of the original amount, and 24 hours were allowed to elapse, to permit matters to find their normal level under the new conditions.

Within six hours of the substitution of the smaller charge, black noses began to form on the tuyeres, and the rate of smelting became decidedly slower. Several empty charges—that is, fuel without ore—were given at intervals; but it became evident, from increasing irregularities, that the furnace was growing cold. A slight addition was made to the fuel charge, and after a considerable number of trials, the normal ratio of fuel to ore for the new conditions was established, and the steady run resumed. A three days' average was taken, as in the former case, and showed the best possible ratio between charge and fuel to be as 8.6 to 1.

The charge was again halved, being now reduced to 370 pounds, and the last-named proportion of fuel maintained until circumstances compelled a change.

In brief, another three days' observation showed a further reduction in the ratio of ore to fuel—7.82 to 1 being the best attainable results. It is also interesting to note that, although

* We now know that the size of the charge is an important element in determining the degree of concentration produced.

great pains were taken to secure the same conditions in every particular during the entire course of the experiment, the matte decreased in tenor with the decrease in the weight of the charge—the average assay reports for the three periods of three days each, beginning with the heaviest charge, being respectively 46.4, 44.5, and 42.1 per cent.—the amount of the same increasing with its poverty in a very nearly corresponding ratio. The slag also (although this may have been a coincidence) showed lower proportions of copper, assaying for the three periods respectively 0.61, 0.47, and 0.41 per cent., which is a greater difference than can be accounted for by the lower grade of the matte, and which in all probability, in common with the latter material, depended upon the more powerful reducing effect, due to the use of thinner charges, and a consequently more perfect mingling of ore and fuel. The capacity fell from 57 tons, in the first instance, to 51 in the second, and down to 41.5 in the third.

The experience at several Arizona furnaces contradicts the above results, quite small charges having been found to answer best, although this may be due to the fact that much of the ore there is fine, while a powerful reducing action is necessary to produce a clean slag.

A proper charge for a 36-inch furnace is from 500 to 800 pounds; while a 42-inch shaft should receive from 1,200 to 1,600, and a 48-inch furnace, 1,800 pounds or more. The large elliptical slag-furnaces at the Lake Refining Works are charged with about 2,600 pounds of ore and flux, experience having shown the advantage of deep layers in the furnace shaft.

As may be imagined, the large rectangular furnaces take still heavier charges, from 3,000 to 4,000 pounds being the ordinary standard.

The shape of the furnace is largely a matter of individual preference, as may be seen by observing the almost equal number of skilled advocates for the round, rectangular, and elliptical form.

Beyond a certain limit, however, the rectangular form alone is used, owing to the feeble penetration of the light blast used in copper smelting.

While the effects of a flaming throat are not so obviously detrimental in copper smelting as in the fusion of the more volatile metals, it still is found by experience that such a condition of affairs is incompatible with the best work, being invariably indicative of a faulty condition of the process.

With an open charge and long-continued high pressure of blast, it is almost impossible to prevent the heat from eventually rising, until the chimney and walls above the charging-door become so hot as to ignite the escaping gases instantaneously.

The ore near the top of the charge soon sinters together; the fuel is largely consumed before it reaches the zone of fusion; the softened lumps of ore stick to the side walls, forming bulky accretions, and the way is paved for the successive steps of "burning out," reduction of metallic iron, and "freezing up," already so frequently alluded to.

While it is sometimes impossible to prevent the early stage of this condition of affairs, when pushing the furnace to its full capacity with a heavy blast, the end results should be borne in mind and the remedy applied in time.

This consists simply in letting the charge sink—under a light blast—until the shaft is empty for a distance of three or four feet below the charging-door. One or more charges of fusible slag are then given, and the furnace rapidly filled full with its normal burden. In this way the overheated walls are cooled, the surface of the charge regains its normal temperature, and the furnace under a few hours of light blast is again ready for a period of hard driving.

In obstinate cases, the cooling of the throat with a spray of water is quite admissible, and often of great benefit.

The question of the characteristics and comparative value of the ordinary fuels used in blast-furnace work has been discussed so exhaustively in most of the standard works on metallurgy as to render it useless to undertake any such task in a treatise like the present, devoted to a certain stated purpose.

The same may be said of fire-brick and other refractory materials, our own domestic brick being quite equal to any of foreign make for all purposes connected with blast-furnace smelting. It is hardly necessary to say that, among the numerous competing varieties of fire-brick, only those should be selected which long and thorough trial has shown to be suited to the purpose; the first cost should have but slight weight in the choice.

TREATMENT OF FINE ORE IN BLAST-FURNACES.

The mechanical condition of the ore to be smelted in blast-furnaces is a matter of scarcely less importance than its chemical constitution.

The evils resulting from an undue proportion of fines are well known.

The formation of an immense quantity of flue-dust is one of the least of these evils, as provision can be made for its collection and reworking, though at an increased cost; but the difficulties resulting from the choking of the furnace, and the sifting of the fine ore through the charge until it pours out in a stream through the tuyere-openings, scarcely altered by its passage through the furnace, are radical, and incompatible with either proper or economical work.

The extent of this evil has encouraged the invention of a great variety of methods for its removal, most of them relating to a consolidation of the fine material into lumps of a suitable size.

The agglomeration of the fine ore in the calcining-furnace has been suggested; but the great expense of fuel and the heavy losses inseparable from a method that, however applicable to such an easily fused substance as silicate of lead, would be usually impracticable when dealing with oxide of iron, render it unnecessary to discuss this practice.

Assuming that the only feasible remedy consists in forming the fine ore into blocks, the experiments executed naturally fall into three divisions:

1. Bricking by the aid of some foreign substance that has the power of holding the ore particles together.
2. Bricking by pressure alone.
3. A combination of the two methods.

The materials tried by the writer and included under the first heading are: silicate of soda (soluble glass), unslacked lime, clay, hydraulic cement, coal-tar and similar substances, sulphate of iron.

In nearly all cases, a certain degree of pressure must be used to form or mold the mixture into the desired shape; this may be obtained by an ordinary brick-machine, or by compressing with the hands, using a mold or not. In No. 2, pressing alone is used. A thorough mixture of the ore with silicate of soda results merely in the coating of each particle with a layer of soluble glass, and in nowise facilitates the agglutination of the ore. On the other hand, when the latter is already compressed into balls or blocks, the dipping of the same into a strong silicate of soda solution is accompanied with great advantage, the surface becoming, on drying, nearly as hard as granite, and effectually preventing any wastage or breakage of the lumps by handling. (It should be mentioned that the circular or oval shape is much preferable to the rectangular, owing to the absence of fragile edges and corners.)

Of course, this material would be far too expensive for anything but the richest ore, sufficient water-glass to thoroughly coat a ton of balls the size of the fist costing, at Eastern wholesale prices, about $3.25.

No substance has been more frequently employed for the purpose indicated than freshly burned lime, which should be slacked with considerable water, and the resulting milk of lime thoroughly incorporated with the ore, until the entire mass possesses the consistency of very thick mortar.

This is usually left in a heap for several days, and then fed into the furnace in the shape of partially dried mud. But much better results are obtained by forming it at once into balls and subjecting it either to the hot sun or to a gentle artificial heat until it is thoroughly dry and hard. The resulting balls are somewhat brittle and fragile, and demand careful manipulation; but are far preferable to the product obtained by leaving it in a heap, and exert a marked effect in the capacity and condition of the smelting-furnace.

The proportion of lime necessary to effect a good result varies greatly, according to the physical condition of the ore, the amount of sulphates present (which form a strongly cohesive cement with the lime), etc., but is usually from 5 to 12 per cent.—less than 5 per cent. seldom producing satisfactory bricks. The cost of mixing alone (lime not included) is from 25 to 40 cents a ton by contract, which sum must be doubled or trebled if it is formed into bricks, depending upon the effectiveness and convenience of the plan. In almost all cases, the addition of lime has a favorable effect upon the subsequent fusion. It is probable that when the water is removed from the lime by the heat of the furnace, the masses again crumble to a certain extent, but not until they have already undergone a certain preparation, which must be of value, to judge from the results obtained in actual work. This method was carried out extensively at the "Gap" nickel mine, Pennsylvania, not only the roasted fines, but also the fine raw pyrites being thus treated, previously to roasting in kilns; the results of the latter process being much better than could be expected from a material possessing such slight cohesive properties as fine granular pyrites.

The Orford Company and many other metallurgical establishments have adopted this method in the handling of finely pulverized, calcined matte, although in most cases the materials are simply mixed into a thick mortar, and charged into the furnace after lying in a heap for a few days. The difficulties and irregularities in the running of the cupola that would certainly result from the employment of an excessive proportion of such unfit material are counteracted to a considerable extent by the addition of a large amount of slag, which serves to loosen the charge and keep everything in normal condition.

In consideration of the advantages already enumerated, and from the fact of its cheapness, general availability, and fluxing qualities, lime may be regarded as the most useful substance yet known for the purpose under consideration, and where bricking under pressure, with subsequent thorough drying, is not attempted.

Clay is also extensively used for the same purpose, and if thoroughly incorporated with the fine ore and allowed to dry for a reasonable time after being made into balls, gives a stronger and less friable product than lime. The quantity added varies from 2 to 5 per cent.

It possesses the serious disadvantage of adding to the siliceous contents of the ore. In the case of calcined matte or highly basic ores, on the contrary, it forms a useful flux. The cheapest variety of clay that possesses the required plasticity should, of course, be selected.

The powerful cohesive qualities of ordinary hydraulic cement long since attracted notice.

Fortified by the favorable opinion of Prof. J. Fraser Torrance, the writer has employed it to brick jewelers' sweeps, and after a long trial is quite satisfied with the results obtained.

He finds about 8 per cent. of cement necessary to produce balls which, after a week's exposure to the air, will bear moderate handling and give good results in the furnace. Of course, the expense of this method forbids its use for ordinary substances.

Where coking coal is available, fine ore can be mixed in large proportions with the coal in the kiln and coked.

Coal-tar and similar substances require the aid of quite powerful compression to answer the required purposes, and have not been found practicable.

A solution of copperas—sulphate of iron—is used in several of the European works to agglomerate fine ore. By careful drying, the balls made with this substance become very hard; but the addition of sulphur to the charge (forming a perceptible increase of matte in cupola work), the very disagreeable effect upon the skin of the operatives, and other minor disadvantages, have prevented its adoption.

The introduction of inexpensive machines for the manufacture of brick from almost dry clay, and capable of exerting an immense pressure, has opened new possibilities to the metallurgist. Although, doubtless, such exist, the author can find no recorded results of bricking fine ore by employing pressure alone, and is therefore obliged to fall back upon some brief trials made under his direction at the Parrot Works, Butte, Montana. The ore used consisted of pyrites concentrates, calcined so thoroughly as to contain only traces of soluble sulphates. The brick-machine used produced about 40 bricks a minute, weighing 5 pounds each, dry, and exerted a pressure of 4 tons per square inch. Under this immense force, the compressed ore slabs already possessed considerable strength, and could be banked up in the usual manner.

Unfortunately, no provision had been made for drying the brick by artificial heat—a most essential part of the process. After a day's exposure to the air, they were smelted in a water-jacket furnace, breaking up to a considerable exent during transportation, but fusing with much greater rapidity and economy than when in a fine condition.

A few that were dried at a gentle heat for six hours became so hard as to bear any reasonable handling, and when broken once in two, were admirably adapted for blast-furnace work. Rapid drying is highly injurious.

The writer is quite convinced of the value of this method, and considers it applicable to any ordinary material.

The essential conditions, after obtaining the proper pressure, are a gentle and sufficiently prolonged temperature, and a sufficient space to dry the necessary quantity. A series of light shelves in a well-ventilated building, heated by steam-pipes, would seem to fulfill these requirements, while the shape and size of the molds could be adapted to the purpose. A round or oval shape is best, thus escaping the wear on sharp corners and angles.

The cost of bricking fine ore in this manner in Montana did not exceed 50 cents a ton; a single machine, requiring 10 horse-power and the labor of eight men, having a capacity of 60 tons in ten hours.

A combination of the two foregoing methods was effected by incorporating a certain proportion of lime or clay with the fine ore, before submitting it to the immense pressure mentioned.

The addition of from 2 to 4 per cent. of either of these substances was accompanied with an increase in the strength and tenacity of the product, and was found especially useful where the process of drying could not be carried out. With proper facilities for a slow but perfect desiccation, no such addition is necessary.

Mr. O. K. Krause, President of the Vermont Copper Company, has obtained excellent results by stirring green fines into the molten slag from his black copper furnaces. From 75 to 100 per cent. of the weight of the slag can be thus stirred in, the cooled mass being broken up and resmelted.

In place of clay, fine slimes from the concentration department or other sources may be substituted, and their metal contents beneficiated at the same time. This practice was adopted by Prof. J. A. Church, at Tombstone, Arizona. An ordinary brick-machine was employed, the cohesive property of the slimes being depended on to bind the fine ore together.

Fine grinding has been lately proposed; it forms a pulp, which becomes tenacious from the minuteness of its particles. This plan is widely practised in England for the balling of the raw Spanish pyrites fines, preparatory to their roasting in kilns. The tenacity generated in this otherwise granular and uncohesive material by a mere grinding is very striking.

Before concluding this subject, the question of smelting fines in their natural condition should be noticed.

While the presence in a cupola smelting charge of even a moderate percentage of fine ore is accompanied with certain evils, such as formation of flue-dust, the choking of the furnace, irregularities in its running, descent of the fine ore unprepared, until it even pours out of the tuyeres, scarcely heated above the temperature of the air, etc., it is a condition that is almost invariably met with to a certain extent, as the mere transportation of ore from one building to another will result in the formation of a certain amount of fines. It becomes important, therefore, to determine at what point the proportion of fines becomes so great as to demand measures for its relief.

This again varies greatly with the quality of the ore, slag, and metal, the power of the blast, size of furnace, capacity and efficiency of dust-chambers, etc.

Here, as in most other instances, no experiments have been recorded to determine this important point, and practice varies with the prejudice or opinion of every individual.

The following experiments were made on ore from the Moose mine, in Park County, Colorado, in 1871, and though relating to the treatment of silver ore, will serve the present purpose as well as though the product had been a copper matte. The furnace was small—2½ by 3 feet—and smelting an exceedingly infusible charge, containing over 40 per cent. of sulphate of baryta and much silica.

The fuel was spruce charcoal, and the blast very weak, causing an extremely slow smelting; but as the conditions remained the same throughout all the experiments, the results possess some value.

The material smelted consisted of ores from the Moose and adjoining mines, from which all that would pass through a 3-mesh screen had been separated, to be submitted to a calcination, the fine ore containing a much greater proportion of sulphides than the coarse, which latter was smelted raw.

To this coarse ore was added a certain amount of carbonate of lead ores, almost free from fines and the requisite quantity of heap-roasted auriferous iron pyrites, to produce a fusible slag. The latter material contained, after roasting, about 25 per cent. of fines; and the mixture as prepared for the furnace, and without the addition of the fine calcined silver ore, carried about 12 per cent. of fines that would pass a 6-mesh screen.

This was regarded as the normal charge, to which, by way of experiment, were added varying proportions of the roasted fines; all other conditions remaining as nearly identical as was practicable throughout the trials.

Representing the quantity of this normal charge that would be smelted in twenty-four hours by 100, the addition of fines produced the following decrease:

10 per cent. fines reduced it to	92
20 " " "	80½
25 " " "	80
30 " " "	64
35 " " "	56
40 " " "	51
50 " " "	42

Aside from the decrease in capacity accompanying the addition of fines, serious irregularities in the running of the furnace were also produced, causing an increase in the cost of smelting per ton, as well as greatly adding to the labor of the men and to the proportion of silver lost in the slag.

An increase in the area of the furnace greatly heightens its capacity for smelting fine ore; and the results obtained in this direction by the use of the large Orford furnace are very striking.

The presence of from 15 to 20 per cent. fines seems to be no drawback at all, when this type of furnace is employed, and even from 50 to 60 per cent. of the charge may consist of this ordinarily unwelcome substance without seriously affecting the running of the furnace, although, of course, its capacity will be somewhat reduced.

After smelting a charge containing a very high proportion of fines for from 24 to 36 hours, it will usually be found that cold, unaltered fines appear at the tuyeres. This results from the constant agitation of the charge by the blast, by which the ore particles are sifted down through the interstices between the fuel and coarse ore until they actually reach the level of the hearth. In such cases the pipes should be removed and all the fine ore within reach raked out of the tuyere-holes with appropriate tools.

All feeding from above should now cease until the charge under a light blast has sunk nearly to the tuyere level, when the shaft should be refilled with alternate layers of fuel and *coarse* ore in the usual proportions, after which the use of fine ore may be resumed. With these precautions, a very large proportion of fine ore may be smelted in this type of furnace without seriously diminishing its capacity or producing any irregularities of importance.

The attention of metallurgists is particularly called to the ease with which raw fine pyrites, technically called "green fines," may be treated in this furnace with the fan-blast. This material often accumulates in great quantities at copper mines, as owing to its mechanical condition it cannot be roasted in heaps, while pecuniary considerations may forbid the erection of an extensive calcining plant for its treatment. Heretofore, when of low grade (from 1½ to 3½ per cent.), it has been either thrown aside in heaps, or allowed to harden and consolidate until it can be broken out in lumps and added to the roast-heaps. By this practice much waste occurs while a large amount of money is constantly tied up in this material.

After experimenting, Mr. J. L. Thomson, of the Orford Company, found that a charge composed of this material and ferruginous slag from the concentration fusion of copper matte could be smelted together to great advantage in the large furnace; the immense volume of blast employed oxidizing the raw pyrites to a considerable extent, and producing a matte of much higher grade than would result from such material under ordinary circumstances. The slag from matte concentration always carries nearly or quite enough copper (from 1 to 2 per cent.), to cover the cost of its re-smelting where fuel is cheap; while the large percentage of protoxide of iron that it carries neutralizes the silica of the green fines, which finds no base in its own composition, all the iron that it contains being combined with sulphur, and consequently unavailable for slag formation.

Where circumstances do not favor the employment of the matte slag, this may be replaced by heap-roasted pyritic ore; in which case, of course, the resulting matte will be somewhat richer.

In any case, the slag resulting from this practice is distinguished by its freedom from copper, owing to the overwhelming amount of low-grade matte present, which cleanses the light siliceous slag to an unprecedented degree. Owing to the large amount of unsatisfied silica in the green ore, the slag is always exceedingly acid, containing from 48 to 55 per cent. SiO_2, and often being so sticky and thick that only the constant and powerful stream of intensely hot, low-grade matte keeps the slag-run open, and prevents the furnace from "sticking up."

The oxidation of the fine sulphide particles by the air-blast carries the heat to the surface of the charge, and produces to a certain extent those evils inseparable from the extension of the high temperature from the zone of fusion to the upper layers of the charge. Owing to these circumstances, a serious burning of the brick walls often takes place, which circumstance, combined with the cutting down of the furnace bottom from the immense quantity of fiery, low-grade metal, favors the practice already recommended of keeping in blast only during twelve hours out of the twenty-four.

The siphon-tap is almost indispensable in this form of smelting, as the quantity of matte produced in a twenty-four hours' run often amounts to 25 or 30 tons.

From six to nine months has been the ordinary length of campaign for these large brick furnaces, running on either a siliceous

oo much coke used – 16% – powerful blast supplies fm
air than is needed by coke – the xcs at a high t
does some pyritic smelting on its own acct. ab
ordinary zone of fusion. – see previous page

or basic slag, at the expiration of which time a week's repairs will again fit them for work.

Not feeling at liberty to give the results obtained in the fusion of green fines at the Orford works, where this practice originated, the author is forced to fall back upon results in his own practice, where the total quantities treated, though much smaller, still aggregate some 30,000 tons. During a four months' campaign, in which a mixture of green fines and matte slag were treated in a large Orford furnace, the average daily (twenty-four hours) results were as follows:

Weight of green fines smelted	49.71 tons.
" " matte slag "	38.40 "
Total	88.11 "
Gas-coke used	16.10 "
Assay of ore	3.87 per cent.
Assay of matte slag	0.94 "
Weight of matte produced	20.67 tons.
Assay of matte produced	10.70 per cent.
Assay of slag produced	0.13 "

Rate of concentration, about 2½ tons of ore into one. The copper produced in this campaign, after taking into consideration the metal gained in the matte slag, and deducting the small amount lost in the slag from the operation itself, agreed almost exactly with the amount calculated from the careful and frequent assays made.

The matte produced, though very low in grade, is roasted in heaps with great facility, and forms a most welcome flux for siliceous ores.

This practice is unique and well worthy of attention.

The size to which ore must be broken for cupola smelting depends upon two factors—its fusibility and its conductivity.

Fusible material, especially if porous—such as ferruginous and calcareous ores, basic slag, etc.—may be charged in fragments from 3 to 12 inches in diameter without producing evil results, although good practice demands its pretty uniform reduction to the size of a large apple (that is, fully 3 inches). The same may be said of fragments of copper matte or metallic substances, which, being excellent conductors of heat, melt all at once, where a piece of quartz or fire-clay might be in a state of fusion on the surface, while hardly heated at the center, and consequently should be invariably reduced to the size of horse-chestnuts, unless smelted in company with a large proportion of basic ore. A striking example of this may be found in the fusion of "mass copper" at Lake Superior, where pieces of this metal weighing five or six tons are smelted on the hearth of a reverberatory furnace with no difficulty or delay, the high conductive power of copper causing an equal distribution of heat and simultaneous fusion of the entire mass. A rock of the same size could never be smelted except by the gradual wearing away of its exterior surface.

BLOWERS AND ACCESSORY BLAST APPARATUS.

All apparatus employed for the production of a blast may be divided into two classes:

I. Those producing a positive blast, and which, if obstructed, must result in the bursting of some part of the apparatus or the stopping of the blower.

II. Centrifugal fan-blowers, which, even if obstructed, continue revolving, consuming much less power than when engaged in actual work, as the air is simply displaced by the vanes, and revolved in the machine itself, without passing out of the pipe.

This distinction is not always clearly appreciated, and serious mistakes in the construction of the plant sometimes arise from a misunderstanding of the properties of the machine which is to furnish the blast.

Such errors can always be avoided by application to a reputable manufacturer of blowing apparatus, as the subject is one to which much attention has been paid by these parties, who are for the most part quite capable of planning and erecting a suitable blowing plant upon a full understanding of the requirements of the case.

Owing to the light blast used by all copper smelters in the United States, the high-priced cylinder blowers as required in iron-smelting plants, are seldom, if ever adopted.

The blowers in almost universal use are frequently styled "fan-blowers" in general; but this is a misnomer; for, although it is true that both types of blower in common use resemble a fan in appearance, the results obtained are widely different; one class belonging to the first section, or "positive pressure blowers," while the other is truly a "centrifugal fan-blower," and belongs under the second heading.

To the positive pressure blowers belong the "Root," "Baker," and "McKenzie" blowers, all of which are too well and favorably known to require description or recommendation.

The volume of air delivered by one of these machines can be calculated with great accuracy, while the pressure simply depends on the rapidity of revolution.

From the nature of the apparatus, the wind is delivered in a succession of rapid, but distinct, impulses and puffs, to which peculiarity the adherents of the apparatus attach a great, although somewhat mysterious, virtue, while its rivals bring forward equally convincing proofs of its damaging effects upon the smelting process.

According to the author's experience with the principal makes of domestic blast machinery, these puffs have neither a damaging nor beneficial effect; an unbroken stream of wind of equal volume and pressure producing exactly similar results.

The machines of the second class, or centrifugal fan-blowers proper, consist merely of a light blast-wheel revolving in an enclosure of much greater diameter, and so shaped that wind enters the escape-pipe largely from the centrifugal force acquired by the enormous velocity imparted to it by the fan.

The smaller blowers make from 4,000 to 5,000 revolutions a minute, and those of even 5 or 6 feet in diameter are speeded up to 1,500 or 2,000.

While this is an excellent device for the delivery of a large volume of wind, the element of pressure is only obtained by a great waste of power, the speed necessary to produce a given pressure increasing out of all proportion to the gain in that quality.

The admirable workmanship of the "Sturtevant" and of other kindred fans, and the skill with which the inherent defects of this method of gaining pressure have been reduced to a minimum, have caused the adoption of this form of blower in many establishments where a considerable pressure is required.

But while the writer fully recognizes its usefulness over a very wide range, its fatal defects as a blower for certain purposes cannot be concealed.

A cupola, for instance, filled to the throat with fine ore and using for fuel a very dense variety of coke, shows evidences of chilling, while the tuyeres become capped with a tenacious slag, which requires a forced blast to keep them open. If the blast is derived from a positive blower, it rises to the occasion, and the pressure of the wind keeps pace with the growing obstruction, as may be seen by observing the gauge. More work is thrown on the engine, and the pressure rises until the obstruction is cleared away, or some portion of the machinery or blast-pipe is overtaxed. (Positive blowers should always be provided with a safety-valve.)

The result is quite different with the fan; for as the wind-stream is obstructed, so is the delivery lessened, until finally, with the complete closure of all blast openings, the work of the engine drops to almost nothing, and the fan revolves at its full speed, neither receiving nor delivering a cubic inch of blast. This may be easily verified by suddenly shutting the main blast-gate between the fan and the furnace. The unaltered stand of the manometer and the sudden decrease of the labor performed by the motive power will sufficiently convince the most skeptical.

With coarse ore, an easily fusible charge, and large tuyere openings, the lightness, compactness, and low first cost may speak in favor of the fan-blower for cupola work.

Elaborate tables, showing the horse-power required under nearly all possible conditions, are issued by the blower manufacturers, and are usually quite correct, as controlled by indicator cards taken in the presence of the writer.

As a guide for possible estimates, it may be assumed that 8 horse-power will drive a positive blower suitable for a 36-inch furnace, while a 48-inch cupola will require from 12 to 14 horse-power.

The speed required by the fan-blowers is so great as to demand some attention to the arrangements of pulleys and shafting to obtain the same.

Care should be taken to use the largest-sized pulleys practicable, both for driving and receiving, and all abrupt belting from very large to very small pulleys will always give trouble.

A much neglected portion of the blowing-plant is the pipe that conveys the blast. By a strict observance of the following rules, much annoyance may be avoided.

Use galvanized iron, No. 22 to 24. For any length of pipe up to 50 feet, make the blast-pipe 20 per cent. larger in diameter than the outlet of the blower; for from 100 feet to 200 feet, 30 per cent. larger, and if the distance be over 200 feet, make the entire pipe 50 per cent. larger. This precaution will diminish friction and greatly increase the effective blast.

If branches are used, remember that, on account of friction, two pipes of a given area can convey only about three-quarters the wind of a single pipe of their combined area.

Make easy curves, avoiding all angles.

All joints should be riveted and soldered. Remember that the slightest leak may reduce the effectiveness of the blast to an enormous extent.

Have tight-fitting blast-gates in the main pipe and at each tuyere.

Maintain a reliable quicksilver manometer connected with the wind-box surrounding the furnace, and accustom your furnace-men to rely upon it as the seaman relies upon the barometer.

Remember, in this connection, however, that a high stand of the manometer may indicate that the tuyeres are obstructed as well as that the blower is working satisfactorily.

A heavy boiler-iron damper should always be fitted in the cupola flue or down-take, and should be lowered to just such a point that the fumes escape lazily, but without issuing from the charging-door.

Through the courtesy of Mr. H. A. Keller, Superintendent, I am enabled to present the average blast-furnace work of one of the great Butte smelters, during the year 1894. It must be understood that the matte from these blast-furnaces (averaging for the year, 49.8 per cent. copper) is blown up to good blister in converters, and shipped East to be treated electrolytically for the notable amount of silver and gold that it contains. Some peculiar features will be noticed in the results, but, while I will not go so far as to say that I believe the practice to be the best that could be adopted in every particular, it must be remembered that the conditions at Butte are also peculiar. Coke, with 20 per cent. ash, costs $11 per ton. Common labor is paid $3 to $3.50 per day. The calcining capacity is nearly always behind the demands of the smelting-furnaces. Coal, with 12 per cent. ash, costs $6 per ton for run of mine. The present Butte cupola practice is the outcome of many years of experience and experiments by men who are skilled metallurgists and thoroughly alive to the importance of small economies per ton on a large tonnage, and any one who sets out to treat the Butte ores with the idea that he can radically cheapen existing methods, is likely to accumulate a large and costly fund of experience before he completes his task.

BLAST FURNACE CHARGE, 1894.

DEBIT.	Gross Lbs.	H_2O %	Net. Lbs.	Insol. Residue.		FeO.		CaO.		S.		Cu.	
				%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
1. Calcines	694	21.0	548	16.7	91.52	47.6	260.85	....	.....	9.0	49.32	13.7	75.08
2. Concentrates, etc.	106	3.0	103	21.5	22.14	38.8	39.96	....	.....	37.1	38.21	9.1	9.37
3. Kiln ores	398	11.0	354	34.0	120.36	39.9	141.25	....	.....	6.5	23.01	9.0	31.86
4. Custom ores	366	3.0	355	46.0	163.30	19.3	68.51	....	.....	25.0	88.75	8.0	28.40
All ore	1,564	....	1,360	....	397.32	....	510.57	....	.....	....	199.29	....	144.71
5. Converter slag	436	....	436	40.3	175.71	51.9	226.28	....	.....	....	......	1.4	6.10
6. Scrap copper	10	....	10	....	......	....	......	....	.....	....	......	90.0	9.00
Ore and slag	2,010	....	1,806	....	573.03	....	736.85	....	.....	....	199.29	....	159.81
		CO_2											
7. Limestone	100	40%	60	2.0	2.00	....	......	52.0	52.00	....	.. ..	....	......
8. Montana coke	198	80%	40	20.0	39.60	....	......	....	.....	...	......	....	......
Total charge	2,308	....	1,906	....	614.63	....	736.85	....	52.00	....	199.29	....	159.81
Less "credits"			492	....	51.84	....	175.70	....	2.55	....	69.44	....	151.43
			1,414	....	562.79	....	561.15	....	49.45	....	129.85	....	8.38
Slag actually made			1,210	46.5	562.65	46.4	561.44	4.1	49.61	....	......	0.3	3.62
Unaccounted for			204	....	0.13	....	0.29	....	0.16	....	129.85	....	4.76
CREDIT.													
1. Matte			237	1.0	2.37	28.3	67.07	....	.....	22.2	52.61	49.8	118.03
2. Flue-dust			255	19.4	49.47	42.6	108.63	1.0	2.55	6.6	16.83	13.1	33.40
Total			492	....	51.84	....	175.70	....	2.55	....	69.44	....	151.43

Slag.

SiO_2	36.2	46.5
Al_2O_3	10.8	
FeO		46.4
CaO		4.1
		97.0

LOSSES IN SMELTING.

Net	— 204 pounds	— 10.6	per cent. of net charge.
S	— 129.85 "	— 65.9	" " "
Cu	— 8.38 "	— 5.2	" " "

The most noticeable feature in this table is the enormous production of flue-dust, amounting to 18.75 per cent. of the ore charged. This arises mainly from the great quantity of fine ore used, owing to the charge consisting of calcines, which are both fine and light; of concentrates and screenings, which are mostly fine; of kiln, or rather, stall-roasted ore, which is exceedingly pulverulent; and of custom ores, which contain a variable, but very large, proportion of fines. Some 60 to 70 per cent. of the ore charged into the cupola is thus in a pulverized condition. This necessitates running with a low ore-column, a circumstance which not only produces much flue-dust, but which keeps the tunnel-head very hot. The result of this combination of circumstances is comparatively slow smelting, low ore-column, and large volume of wind, and this produces the oxidizing effects so often and strongly insisted upon throughout this work. As may be expected, the oxidation is tremendous, no less than 66 per cent. of the sulphur in the charge being removed during this fusion. It will be noticed that all the ores treated are comparatively high in sulphur for blast-furnace work, especially when we consider that a 50 per cent. matte is to be made from ores averaging only 10.64 per cent. copper and yet carrying 14.6 per cent. sulphur. Thus it becomes essential to remove most of the sulphur in the blast-furnace. The assay of the slag is extraordinarily low, only 0.3 per cent. copper, and I should be inclined to believe that this accounts for at least a portion of the missing copper, though the amount of the latter is not large, being only about 5 per cent. of the total copper in the charge, or scarcely more than one-half a unit. The ore is remarkably free from injurious impurities.

COST OF SMELTING IN WATER-JACKETED BLAST-FURNACES.

It is difficult to make a general estimate of the costs of running a blast-furnace. Even after omitting the very variable item of "General Expenses," (which includes management, other superintendence, office expenses, laboratory, insurance, taxes, etc.), it is impossible to offer figures that shall have any universal application. This is true for three reasons:

1. Because prices of fuel, labor, supplies, freights, etc., vary in different localities by several hundred per cent. And, even when these items are reduced to a common standard for purposes of comparison, they will be more or less invalidated by differences in the quality of the fuel and labor, local customs as to what constitutes a day's work, etc.

2. Because the arrangement of the works as regards handling fuel, ores, and products; the presence of abundant water to granulate and remove the slag, or of a convenient, terraced piece of ground on which to erect the works, will make a great difference in the expenses.

3. Because, as already explained, there are two totally different kinds of smelting done in copper blast-furnaces, one being the simple fusion of the ores; the other, combining a certain degree of oxidation (and, consequently, concentration) with this fusion.

As this is usually a considerably slower operation than the mere fusion of the ore, it might easily happen that its costs, per ton of ore smelted, might be higher, though the operation would have produced a much greater profit to the smelter than the quicker and cheaper one.

There will also be considerable expenses for re-treating flue-dust in works using a heavy blast, or where a large proportion of fines is treated, and many other local circumstances that prevent a fair comparison of results.

The following figures are offered as a sort of compromise estimate, the furnace (36 inches by 132 inches) smelting 100 tons of ore per 24 hours, besides its own flue-dust and foul slag, and steering a middle course between simple fusion and extreme oxidation.

The plant is supposed to be built upon a side hill, so that all materials can be brought to, and taken from, any portion of the works by an ordinary steam railroad.

The furnace is charged direct from hand-barrows, the shovel being only used for slightly leveling the charge. The matte is tapped into molds and loaded direct on to the railway cars at the furnace. The cost of delivering the original ores into the furnace bins is not included in this estimate.

The prices of coal and labor are assumed for the occasion, these items alone being fictitious. All other work and results are taken from actual practice for long periods.

The ore is a mixture of roasted cupriferous pyrites, manganous silver ore, and siliceous ores carrying some silver, gold, and copper.

The average smelting mixture contains about 5½ per cent. copper, and $13 in silver, and $4.50 in gold per ton. The matte produced averages about 40 per cent. copper, and $100 in silver and $33 in gold per ton. The slag averages 0.45 per cent. copper, a trace in gold, and below one ounce (0.0034 per cent.) silver per ton.

There are two cupolas in constant operation. No superintendence, assaying, or other general expenses are charged in this estimate, as their amount depends largely upon how much other work is going on at the same smelter that may share these expenses. The water for the boilers, jackets, and granulation and removal of slag is brought from a neighboring stream by a high-level ditch.

COSTS OF RUNNING BLAST-FURNACE PER 24 HOURS.

Labor (per shift of 12 hours)—	
½ foreman at $5.00	$2.50
2 men on charging floor at $2.50	5.00
1 furnace-man at $3.00	3.00
1 helper at $2.25	2.25
1 laborer on matte and flue-dust at $2.00	2.00
½ laborer at slag sluice and in yard at $2.00	1.00
¼ blacksmith and helper at $4.50	1.13
Total labor for 12 hours	$16.88
Total labor for 24 hours	33.76
Fuel (for 24 hours)—	
14.7 tons coke at $6.50	$95.55
Coke, wood, etc., for drying forehearths, etc	0.38
Total	$95.93
Blast (for 24 hours)—	
2.6 tons coal at $4.25	10.05
2 engineers at $3.50	7.00
1 laborer at $2.00	2.00
Oil, lights, and miscellaneous	2.17
Total blast for 24 hours	$21.22
Supplies, repairs, and renewals (for 24 hours)—	
Oil and small stores	$0.76
Renewing tools and charging-barrows	1.98
Renewing steel matte molds	0.72
Sand and clay	0.33
Renewing forehearth every three weeks	1.64
Repairs on furnace	1.82
Repairs on engine, blower, pipes, etc	0.88
Lighting building and yard	1.96
½ machinist at $3.50	1.75
Sinking fund to replace furnace, etc	2.67
Proportion of blowing-in and out	0.44
Repairs on slag-launder	1.77
Shifting discharge end of slag-launder	0.85
Unclassified expenses	0.80
Total supplies, repairs, and renewals for 24 hours	$18.37
Retreating flue-dust (for 24 hours)—	
Recovering and bricking, 5.2 tons of flue-dust at $0.88	$4.58
Lime	1.86
Power and repairs on hydraulic brick-press	3.22
Transporting bricked flue-dust to ore-bins	1.28
Resmelting 5.75 tons bricked flue-dust at $2.00	11.50
Total retreating flue-dust for 24 hours	$22.44

Retreating foul slag (for 24 hours)—	
Returning 0.87 tons foul slag to ore bins	$0.22
Resmelting same at $2.00 per ton	1.74
Total retreating foul slag for 24 hours	$1.96

Résumé of totals.	Per 24 hours.	Per ton ore.
Labor	$33.76	$0.33.7
Fuel	95.93	0 95.9
Blast	21.22	0.21.2
Supplies, repairs, and renewals	18.37	0.18.4
Retreating flue-dust	22.44	0.22.4
Retreating foul slag	1.96	0.02
Miscellaneous trifles	8.88	0.09
Grand total	$202.56	$2.02.6

The above estimate represents good practice on favorable ores and with moderately cheap labor and fuel, but is reached, and excelled, in several plants in this country, where exceptional care and skill is employed.

CHAPTER XIV.

PYRITIC SMELTING.

By "Pyritic Smelting" I mean, *The fusion of sulphide ores by the heat generated by their own oxidation, and without the aid of extraneous heat, such as carbonaceous fuel, the electric arc, etc.*

This is the plain definition of "Pyritic Smelting" as understood by American metallurgists. It is a term that has apparently come to stay, and must in nowise be confounded with the mere employment of sulphides to collect the valable metals in a matte, where no intentional use is made of the heat generated by their oxidation. (Kongsberg, Sala, etc.) This latter operation is simply a matte smelting, and the accidental circumstance that much of the pyrites was melted in a raw condition, instead of being roasted, was due to the belief of the older metallurgists that a fall of 33 per cent. to 50 per cent. of matte was necessary for the thorough extraction of the precious metals; an error classically perpetuated by Kerl, in a form that will, no doubt, remain durable for some years to come.

I need hardly say to metallurgists in this country, that this is an entire mistake, and that neither 50 per cent., nor yet 5 per cent., of a matte fall is necessary for the practically perfect extraction of the gold, silver, and copper, providing the slag be suitable. (The Norwegian ores were eminently suited to make a clean slag.)

At the present stage of pyritic smelting, it is not absolutely correct to say that *no* heat is received from extraneous sources.

A hot blast is frequently employed, as it produces a beneficial effect entirely out of proportion to the number of heat units it carries into the furnace. A small percentage of coke (1 to 3 per cent.) is also generally added, in the present tentative and imperfect state of the process, though runs of six days have been made absolutely without coke, when the ores were suitable.

But hot blast is universally employed in iron smelting; is frequently resorted to in calciners and reverberatories, and would be gladly welcomed by many experienced copper metallurgists for ordinary blast-furnace work, were it not for the fear of reducing iron to the metallic state.

Again, it is not many years since we were compelled to throw a few sticks of wood into the copper bessemer converters toward the close of each blow, that there might be sufficient heat to make the copper very fluid for pouring. At present, with larger converters, quicker work, and generally improved conditions, we use no carbonaceous fuel, and, indeed, often have too high a temperature.

In the same way, and when the ores are favorable, we may expect to practise pyritic smelting without the addition of any carbonaceous fuel.

This process is attracting much attention in the United States at present, and no work on the metallurgy of copper would be complete without a full discussion of the matter. Excepting a few brief pamphlets by the practical introducer of the process, Mr. W. Lawrence Austin, and a certain amount of discussion in journals and transactions of societies, there is no literature on the subject.*

This must be my apology for going into the question at some length, and for reviewing nearly all the work that has been done in this line up to the present time.

Before adopting a new method for the treatment of his ores, the metallurgist naturally asks first:

How much can I save by it?

What will be the cost of the plant?

In most places, the great expense of smelting ores is the fuel. To arrive at an approximate idea of the proportion that it bears to the entire cost of the smelting operation, I have constructed a table, showing the various items that make up the cost of smelting copper ores in blast-furnaces in this country, and to a limited extent in Canada, where the conditions are much the same as in certain portions of the United States.

The first five instances are from smelting done at different works more or less directly under my own superintendence, and I can vouch for their accuracy. The remaining examples represent the conditions at other smelting works with which I am personally familiar, and are very close to the truth. The wide variations

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* Since writing the above, Mr. Herbert Lang has published a most valuable paper on this subject that will be referred to later.

in the cost of fuel, labor, supplies, etc., are caused by many different circumstances, such as the varying prices in different localities, the greater or less fusibility of the ores, the mechanical condition of the ores (whether coarse or fine), quality of fuel, technical skill and economy, arrangement and magnitude of plant, sizes of furnaces, height above sea level, etc. In order to put all the examples as nearly as possible on the same basis, I have counted limestone, manganese, or iron-flux as though it were ore. For it costs as much to smelt a ton of flux as a ton of ore, and it is simply a fortuitous accident if ores are self-fluxing, or can be made so by mechanical concentration. Administrative and other general expenses are omitted, as averaging about the same in large, well-managed plants. The amount of ore given in the table as smelted per day, is the average capacity of one blast-furnace at the works cited, but in almost every instance there are two or more furnaces in operation, which tends to reduce the cost of smelting, per ton of ore.

COST PER TON (2000 LBS.) SMELTED, IN DOLLARS AND CENTS.

Smelter.	Capacity of 1 Furnace in Tons Per 24 Hours.	Coke. Per Cent.	Labor.	Blast.	Supplies.	Repairs, Etc.	Total Cost Except Fuel.	Fuel.	Grand Total.	Per Cent Cost of Coke.
A........	110	14	$0.28	$0.17	$0.12	$0.10	$0.67	$0.91½	$1.58½	58
B........	95	13	0.46	0.25½	0.09½	0.26	1.07	0.65	1.72	38
C........	76	16⅔	0.47	0.29	0.18	0.13	1.07	2.17	3.24	67
D........	115	16⅜	0.38	0.19	0.11	0.30	0.98	1.24	2.22	56
E........	65	17	0.71	0.51	0.43	0.39	2.04	6.10	8.14	75
F........	45	16	0.66	0.28	0.22	0.31	1.47	2.96	4.43	61
G........	50	15	1.22	0.41	0.56	0.77	2.96	5.23	8.19	77
H........	72	14	1.03	0.38	0.45	0.38	2.24	4.40	6.64	66
Average	....	15½	....	....	....	....	....	......	....	60.5

[It must be remembered that the price of coke in the above estimates varies from $2 per ton, at certain points in the Middle States, to over $60 per ton in inaccessible portions of Arizona. There is nothing to be learned from the averages of the other separate items, as the conditions and prices of labor and material vary too much to afford any correct conclusions. But the proportionate cost of the coke to the entire cost of smelting is exceedingly instructive, and it is for this object that the table was constructed.]

We therefore see that in large works, well constructed, and with good modern arrangements for the handling of ores and products, the coke used in smelting the charge forms about 60 per cent. of the cost of the blast-furnace process. No other item, or group of items, presents such a large margin for reduction, and nowhere else is it possible to effect a lessening of costs that can in any way compare with this.

I am informed by those who have had the most experience in running pyritic smelters on a commercial scale, that 2 per cent. of coke is an ample proportion to use, when smelting ores in any way suited to the process. My own small experience confirms this statement, and very recent information from English friends who are experimenting with it, agrees with the results obtained in this country. But to allow liberally, I will add two-thirds to this estimate, and assume that the present pyritic smelter will use 2 per cent. plus (66.66 per cent. of 2 per cent.) equal to 3.33 per cent. of coke.

In ordinary copper smelting in the United States, the average consumption of coke in the blast-furnace process will fully reach one-sixth of the weight of the ore, or 16.66 per cent. Many smelters claim to be doing better than this; but when the weight of the fluxes is considered as ore, and when the consumption of coke is based on the figures derived from the entire amount of coke that was paid for during the year, and not on the proportion weighed out for each furnace charge, it will not be found that my estimate is too high. The unavoidable loss in fines, increased by the railway transportation for long distances, adds largely to the percentage of this material employed, and the inevitable deficit that appears when stock is taken, is an uncertain, but important factor.

Assuming, therefore, that our coke bill is two-thirds of our total smelting costs, that we are using $16\frac{2}{3}$ per cent. of coke in smelting, and that by employing pyritic smelting, we shall require only 3.33 per cent., or one-fifth as much as by the ordinary method; we find that under these conditions, our coke bill will be only 20 per cent. instead of 60 per cent. of our total expense of smelting. Thus we save 40 per cent. of the entire cost of the blast-furnace work, or an amount equal to the aggregate of all the other items of labor, blast, supplies, and repairs.

This is such an extraordinary and unprecedented economy, that its adoption would be forced upon the entire copper-smelting world within a very short time, were it attainable with certainty, and unaccompanied by too many serious drawbacks.

Its advantages and disadvantages will be enumerated in the succeeding pages, and the reasons considered why it still plays such an unimportant part in the domain of commercial metallurgy.

It is somewhat difficult to treat of Pyritic Smelting in the brief space that can be afforded it in this work, as at least two separate and distinct methods are embraced under this title, and these again require subdivision.

1. Pyritic smelting with column charging.
(*a*) For copper ores.
(*b*) For precious metal ore with but little copper.
2. Pyritic smelting with layer charging.
(*a*) For copper ores.
(*b*) For precious metal ores with but little copper.

As certain principles and reactions are common to all these varieties of pyritic smelting, it will be more convenient to consider the subject broadly at first, and later to take up the variations in detail.

We already understand that the pyritic smelter derives most of the heat essential to the fusion of his ores from the oxidation of the sulphur and iron of the ores themselves.*

But this is not the only striking advantage peculiar to this method.

In treating ordinary sulphide ores, the largest and most expensive portion of the plant consists of the calcining department, in which, with a serious expenditure of labor, fuel, and time, we are at great pains to burn and destroy, more or less perfectly, the very sulphur and iron that form nature's fuel to melt the ore itself. In other words, we employ extraneous fuel to burn up the fuel in the ore, that is already sufficient for its treatment.

It is as though we employed the contents of our coal-bins to burn up a large portion of our coke-pile, so that we could get at the residue of it more conveniently.

This operation of roasting, or calcination, costs from one-half to one-sixth as much as does the smelting itself, and has for its main result the destruction of the natural source of heat contained in the ore.

A simple theoretical calculation of the heat derived from the oxidation of the sulphur and iron contained in pyritous ores shows that it is sufficient to thoroughly melt the ore and leave a considerable margin besides.†

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* The oxidation of any arsenic, antimony, tellurium, tin, zinc, lead, etc., that may be present in the ore, also furnishes heat, but rarely in sufficient amount to warrant consideration in a purely practical treatise.

† It may be asked by non-professional readers why this high temperature is not generated in the calcining furnace, where the sulphur and iron are oxidized as thoroughly as in the pyritic smelter. It is quite true that exactly the same

Hence, the second great advantage claimed by the pyritic smelter is the saving in investment of the first cost of a calcining plant, and the saving in running expenses of the cost of calcining the ore. Two handlings of the ore are also saved, as it may be dumped direct from the mine-cars into the furnace-bins.

Assuming that an ore-mixture contains the requisite amount of iron pyrites to produce the necessary temperature, that enough silica is present to form a proper slag with the iron after it is oxidized, and that there is enough copper in the ore to collect the gold and silver, we know that the following phenomena will occur in pyritic smelting.

The water held mechanically by the ore will be driven off in the extreme upper portion of the furnace. Before a red heat is reached, and consequently but a few feet below the charging door, the first atom of sulphur belonging to the iron pyrites (FeS_2) will be volatilized, a portion of it being sublimed direct as metallic sulphur, while a further portion is burned to sulphurous and sulphuric acids.

As the charge sinks and the heat increases, the sulphides begin to soften and their component parts to be disassociated and oxidized. The FeS has its remaining atom of sulphur burned to SO_2 and SO_4, furnishing a large amount of heat by its combustion, while a still higher temperature is generated by the burning of the iron to ferrous oxide, in which condition it combines with the silica present to form a slag.

The atmosphere of the true pyritic smelter is oxidizing. This is self-evident, for if it were not so, the charge would be simply melted down as pig iron is melted in the foundry cupola; and the absence of an oxidizing atmosphere would stop the combustion of the sulphur and iron, so that there would be no source of heat. Yet the reactions between the sulphides and oxides cause a momentary reduction of the copper to the metallic state, in which condition it instantly combines with sulphur to form a subsulphide. There should be, also, a sufficient amount of ferrous sulphide escape oxidation to dilute the subsulphide of copper to a matte of

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number of heat units are produced in the one furnace as in the other; but in the pyritic furnace the oxidation is almost instantaneous, liberating a vast amount of heat in a few moments, instead of distributing it evenly over many hours as in the calcining process. A parallel case may be found in any heating stove. If filled with coal, and the dampers left wide open, it will quickly produce an intense heat; but with closed dampers, the same quantity of coal will furnish a gentle heat during the entire day.

the proper grade, this matte, of course, acting as a gatherer of the gold and silver in the ore.

By smelting slowly in a suitably shaped furnace and with a large volume of wind, pretty much all the sulphur and iron can be burned, and the copper obtained in a metallic condition, or even oxidized and carried into the slag.*

Neither of these results is desired, and it requires constant watchfulness to so manage the pyritic smelter that the desired degree of concentration is attained, without making the matte so rich as to cause excessive losses in the slag.

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* Witness the process at Ely, Vermont, where matte, thrice roasted in kilns, and still carrying 8 per cent. sulphur, is smelted with 17 per cent. coke directly for black copper, containing only 2 per cent. iron, and but a fraction of a per cent. of sulphur. A small addition of siliceous material is made, and a large volume of wind, at very low pressure, employed. If the matte were melted down rapidly as in ordinary blast-furnace work, the resulting product would be a fair blue metal (56 per cent. copper), as I have proved more than once by direct experiment. But these low furnaces with perpendicular walls and an enormous volume of wind at very low pressure, exert a powerful oxidizing influence upon the charge, by which most of the sulphur is volatilized and the iron oxidized and slagged. Two-thirds of the copper is obtained direct as black copper of remarkable purity, the other third appearing as high white metal (76 per cent. copper), which is also kiln-roasted and resmelted with the lower matte. On starting a new furnace, a considerably smaller portion of the copper is obtained in the metallic form than later, when the walls in the neighborhood of the tuyeres are eaten into deeply. On the other hand, if the smelting zone is narrowed by the formation of wall accretions, but little metallic copper is obtained, the reducing action of the furnace being greatly increased by the artificial boshes thus formed. Any increase in the pressure of the blast, and consequent rapidity of the smelting operation, is always followed by an increased production of low-grade matte. This phenomenon was exhibited in an instructive, as well as amusing manner, many years ago, when, on the arrival of some of the important stockholders of the company, the business manager directed the furnace foreman to charge back several bars of metallic copper into each furnace, and to increase the coke and blast; expecting to thus be able to show a row of copper furnaces running at great capacity, and giving a heavy yield of metal when the hour for tapping arrived. At the appointed time he brought his visitors to the furnaces, and after pointing out the great increase in capacity that was shown by the heavy slagfall, he ordered the men to ladle the copper. But instead of the 8 or 10 200-pound bars of black copper, there were some tons of very fair blue metal, and but a few thin slabs of metallic copper at the bottom of the first two or three pigs. The sulphur in the charge not being oxidized during this rapid smelting, had not only made a great quantity of low grade matte, but had "thrown back" the metallic copper into a sulphide again.

Other things being equal, and with the proper volume and pressure of blast, the degree of oxidation, and consequent concentration, depends largely upon the shape of the furnace.

A contraction of the furnace at the tuyeres causes a higher temperature, more rapid driving, a more powerful reduction, and, consequently, a greater production of low-grade matte. It also naturally causes a more acid slag, and one freer from valuable metals. For the unoxidized iron is now combining with the unburned sulphur to form a great quantity of highly ferruginous matte, while the slag is far more siliceous for lack of this very iron; and being more siliceous, has a lower specific gravity, and is freer from copper, gold, and silver. For the values in blast-furnace slags proceed nearly always from grains of matte that are mechanically entangled, or suspended, therein, and less from oxide of copper combined with the silica of the slag.

This slag, being more siliceous, requires a higher temperature for its formation and fusion, and as less sulphur and iron are being oxidized, and thus less heat generated, instead of more, it might be supposed that the furnace would chill. But I have not found this to necessarily be the case. There is, on the whole, less heat produced than before throughout the entire column of ore in the furnace; but the combustion is now concentrated in a small zone in front of the tuyeres, and, instead of a gentle red heat, which gradually burns a large portion of the sulphur and iron as they pass slowly through the furnace-shaft, we have a fierce heat, limited to the zone of fusion, and probably producing hotter slags than before. The large fall of low-grade matte also tends to keep the hearth hot and ensure rapid driving.

There is very little gained, however, by such practice. It is simply a melting-down of the sulphides, and a removal of the earthy portions of the ore, and a very small part of the iron. The main advantage of pyritic smelting is missed, *i. e.*, the roasting influence, by which the sulphur is burned to a gas and most of the iron oxidized and slagged, so that a high-grade matte is produced in one operation.

Not many years ago, I cracked several cast-iron jackets belonging to a copper furnace, and was obliged to temporarily substitute some boshed water-jackets from an old lead furnace, thus narrowing my furnace, at the tuyeres, from 42 inches to 32 inches. Although not using the genuine pyritic smelting process, I was running on badly roasted pyrites, and getting so much benefit from the combustion of the sulphur and iron in the same, that only about 8 per cent. of coke was needed for smetling the charge. I was obtaining a good matte, the concentration being about 8 to 1 on a 6 per cent. ore, and the resulting slag was very fluid and basic. After the hearth was narrowed, and with exactly the same charge and conditions, the rate of concentration dropped, and remained, at 3½ to 1, the slag became nearly a bisilicate; yet the heavy fall of low-grade, fiery matte kept the hearth so clean and open that the furnace ran on the same amount of coke as when the normal charge was being smelted. The large Raschette furnaces, introduced by the Orford Company, long before the days of pyritic smelting, are powerful oxidizers, their long, narrow shaft, vertical walls, and great volume of blast, introduced through numerous tuyeres of unusual diameter, all tending toward the one end. They were not water-jacketed (though now slightly cooled with water circulating through pipes buried in the brick-work), and while running such a furnace on Ely, Vermont, pyrites, I found that as the walls burned out at, and above, the tuyere level, the matte became richer, though the smelting was slower, and it took more wind, and a little more coke, to keep the furnace at the proper temperature.

This shape of the furnace-shaft is one of the most important of all influences that determine the degree of reduction or oxidation that shall be exerted upon the ore smelted therein.

The importance, therefore, of keeping the hearth and tuyeres free from heavy crusts and wall accretions becomes very evident. If these cannot be easily barred away from the charging-floor, it is much more profitable to blow out the furnace and clean it out, than to continue making a lot of low-grade matte for even 24 hours. A well-arranged water-jacket can be blown out, barred out, and blown in again with a new forehearth, in four hours, and the cost of doing this is very trifling compared with the loss occasioned by the lowering of the grade of the matte, due to the artificial boshes that have been formed in the furnace. One of the great advantages of employing water-jackets is, because the crusts are barred off their cool, smooth, inner walls so easily.

Even a slight diminution in the size of the shaft from top to bottom is useless, and indeed distinctly disadvantageous, where an oxidizing action and a matte of good grade are desired. And it must be a very anomalous condition of things where these results are not earnestly wished for in copper smelting.

A large furnace is much more favorable to an oxidizing smelting than a small one, as will be more fully shown when we come to consider "Pyritic Smelting with Layer Charging."

The next most important factor in determining the amount of oxidizing influence in a pyritic furnace is the size of the charges. Light charges favor reduction; heavy charges, oxidation.

In lead smelters, a powerful reduction is required, and this has led to an intimate mixing of the ore and fuel by the use of exceedingly small charges of each. In ordinary copper smelting, where no such reducing action is wanted, a higher grade matte will be obtained, and a more regular action of the furnace induced, if these layers of ore and fuel are largely increased in bulk. In true pyritic smelting, with a minimum of coke, this point becomes of less importance; but where from 5 per cent. to 16 per cent. of coke is used, the charge should be as heavy as the furnace can bear.

In a vertical furnace, 6 by 96 inches in size, I have been accustomed to use 5,000 pounds of ore, 50 per cent. of it being coarse, as the unit of charge, varying the coke as occasion demanded, and using from 0 to 1 per cent. of slag, according to the carelessness of the workmen in making foul slag.*

For a round furnace, 42 inches in diameter, an ore charge of about 1,500 pounds will be right. In a general way, I have found that for medium furnaces of say 15 square feet area, a burden of 200 pounds to the square foot is about the most advantageous. This must be lowered to some 150 pounds per square foot for much smaller furnaces, and increased to 225 pounds or 250 pounds for the largest ones.

The size of the pieces of fuel also influences the process of oxidation in the blast-furnace to a greater extent than is usually believed. Small coke yields a lower grade matte, while coarse coke favors oxidation and concentration, and also permits the use of heavier charges.

It is hardly necessary to point out that the use of coke at all, in true pyritic smelting, works exactly against the conditions and reactions that we are trying to bring about. It uses up the oxygen of the blast, forming carbonic oxide and producing a powerful reducing atmosphere, and prevents the oxidation of the sulphur and iron which should form the main source of heat. It is only by an ex-

* At a large furnace, using a good ore-mixture, and provided with proper settling appliances, one-half of one per cent. of foul slag is as much as should ever be made during normal running.

cess of blast that the atmosphere of the smelting zone can be kept at all oxidizing, and a very slight increase in the amount of coke used is quickly followed by a lessening in the concentration effects, and an increased fall of low-grade matte.

In enumerating the main factors that influence the oxidizing and reducing action of the furnace, I have not referred particularly to the blast, as it is self evident that an increased blast will heighten the oxidizing powers of the furnace, and vice versa. But there are important points connected especially with the volume and pressure of the blast, that will be considered later.

To sum up: Oxidation of the charge, and consequent raising of the grade of the matte, is favored by:

1. Large area of furnace at tuyeres, and, consequently, perpendicular walls. (No boshes.)
2. Large furnaces, which means rectangular furnaces, as the circular ones cannot well go beyond 42, or better, 36 inches, with the low pressure of blast suited to this work.
3. Heavy charges in the furnace; *i. e.*, thick layers.
4. Fuel in comparatively large pieces.

It must be remembered that the blast-furnace is a most capable and economical apparatus for the calcination of pyritic ores during smelting, providing that we can replace the coke entirely, or to a considerable extent, by the sulphur and iron of the ore itself.

PYRITIC SMELTING WITH COLUMN CHARGING.

Those familiar with Pyritic Smelting are necessarily aware that in thus treating heavy sulphide ore, two of the greatest difficulties are:

(*a*) The agglutination of the charge in the upper portion of the furnace, in consequence of the tendency of the heat to creep upward, and, to a lesser extent, from the volatilization of the first atom of the sulphur contained in the iron pyrites, in a metallic and sticky condition. (On very pyritic ores, there will be some 7 tons per 12 hours of this metallic sulphur.)

(*b*) The packing and settling together of the charge, due to the large proportion of smalls usually present, and frequently much aggravated by the tendency of iron pyrites to decrepitate when exposed to a sudden heat.

These difficulties are often serious, and may cause grave irregularities in the running of the furnace.

Mr. W. Lawrence Austin, of Denver, Colorado, is fully entitled to the credit of carrying out to a commercial success the principles that John Hollway of London proved to be practicable, and has received patents in various countries for an invention intended to obviate these annoyances.*

Austin's original apparatus is, in the main, an ordinary, water-jacketed copper-blast-furnace, which has suspended in the vertical axis of its shaft, and extending downward to within 14 to 20 inches of the tuyere level, a water-jacketed tube. This tube has the same shape as the furnace itself, but being considerably smaller than the latter, an annular space is left between it and the walls of the furnace. The pyritic ores and concentrates are fed into this central tube, while the siliceous ores, fluxes, slags, etc., are charged into the annular space surrounding the tube. The gases are drawn off from this annular space, as in many of the lead-blast-furnaces, the materials fed therein being coarse and easily permeable by the draught.

In the tube, however, the ore column is generally very dense, owing to the usual fine condition of pyritic material, and to the decrepitation of much of it that is coarse. A hot blast is used, and this meets the sulphides just as they enter the upper regions of the smelting zone. All metallurgists are well aware of the great effect produced by even a slight increase of temperature on the rapidity and energy of chemical reactions, but no one who has not witnessed it would credit the results that are obtained by merely heating the blast to 800 or 1,000 degrees Fahr. (427 to 538 degrees Cent.)

The oxidation of the sulphur and iron is almost instantaneous, and the resulting temperature is far beyond what we are accustomed to in a copper furnace. Bisilicates are rendered as fluid as are monosilicates in a common furnace, and even trisilicates are thoroughly fused, so that they will flow with ease.

For this especial kind of smelting, Austin recommends small, but numerous, tuyeres and a high pressure of blast. The heat does not extend up the central tube, the draught being rather downward than upward in it; consequently none of the combustible portions of the ore are wasted, and the heated oxygen coming

* It is proper to add that no furnaces of the peculiar make alluded to as designed by Mr. Austin are in operation, he having, after a longer experience, considerably modified his original ideas. But a brief glance at his first method of construction will enable us to more fully appreciate the difficulties that are to be encountered and overcome in this operation.

into intimate contact with the melting pyrites, combines with a large portion of the sulphur and iron with violence, fusing the ore, together with the silica from the annular space, which now mixes with it, almost instantaneously. The molten drops of sulphide, as they fall in front of the tuyeres, are shattered and comminuted by the strong blast, and actually bessemerized, the high temperature and immediate contact with air disassociating and oxidizing the sulphur and iron in an instant. The slags are exceedingly hot and liquid, and being highly siliceous from the large amount of dry, quartzose ores that can be added to the charge, are very free from enclosed grains of matte.

Lead and zinc are largely volatilized as an oxide, though a considerable proportion of the latter enters the slag.

Copper, when not present in too large amounts, continues to asserts its affinity for sulphur, and is mostly carried into the matte.

But as this method, or rather a modification of it, has been mainly used for gold and silver, and in districts where copper ores were very scarce, we lack information regarding the results that would be obtained in trying to make a rich copper-matte.

Mr. Austin, the inventor, informs me that he has never made a matte running higher than 15 per cent. in copper, although his concentration might be 10 or 15 into one, owing to the very small quantity of copper in the original ore. Owing to the powerful oxidizing atmosphere in front of the tuyeres, we cannot feel certain that for a pure copper proposition, such as making a 40 or 50 per cent. matte from 5 per cent. pyritous ores, there might not be danger of too great a loss of copper in the slags. The same remark will apply to nickel and cobalt in a still higher degree, they being even more easily oxidized than copper.

It is to be hoped that this important point will be settled before long, as few methods hold out greater inducements for at least an experimental trial at some of the greatest deposits of cupriferous pyrites, where the ore is not suitable, or there is not a market for the sulphur. It is peculiarly adapted to pyrrhotite ores, as the annoyance arising from the volatile atom of sulphur in ordinary pyrites, will here disappear.

It is also possible that the fumes from this process may be utilized for the manufacture of sulphuric acid. Any volatilized, metallic sulphur that they contained could be burned to sulphurous acid outside the furnace, and the gain would be a double one; as the amount of sulphur thrown into the atmosphere from a pyritic smelter will have a harmful effect on surrounding vegetation. It must, however, be remembered that just in those localities where there is no valuable vegetation to be damaged, sulphuric acid itself would often be a valueless product.

The gases from a pyritic smelter running on heavy pyrites ores consist mainly of nitrogen and volatilized sulphur and sulphurous acid. Zinc, lead, or other volatile metals present, would have to be removed before the gases entered the acid chambers.

Mr. Austin has sometimes experienced a difficulty from the oxidation of the iron of the pyrites to *ferric* oxide, instead of to *ferrous* oxide. This occurs directly in front of the tuyeres, and produces an infusible substance, loath to enter the slag and very refractory when it does combine with silica, and prone to ally itself with the matte in the shape of a magnetic oxide. He suggests the injection into the tuyeres of oil, gas, or other reducing agents, to prevent the formation of this unwelcome and refractory oxide.

I cannot understand the advantage of this plan; for the whole process being based upon the maintenance of a powerfully oxidizing atmosphere in the furnace, it does not seem rational to attempt to neutralize a portion of the oxygen of the blast, by introducing into it substances that will waste its effect.

The capacity of this pattern of pyritic furnace is very great. Mr. Austin claims that a furnace of medium size, that might have a daily capacity of 50 tons in ordinary lead and copper smelting, may be run up to 200 tons, when equipped and managed as just described. Nor is this rapid smelting necessarily accompanied by the lowering of the matte grade, as is ordinarily the case. The bessemerizing influence of the powerful blast seems to be sufficient to accomplish the necessary amount of oxidation, even when driven at the most rapid rate, especially if it be possible to avoid the use of carbonaceous fuel.

Another reason for the greater capacity of pyritic furnaces is the lessened bulk of the charge, the ore containing its fuel within itself, and requiring little or no flux, owing to the wide variety of slags that are possible at the high temperatures attained.

The amount of flue-dust is not so great as in ordinary smelting, owing to the lessened volume of gases that escape from the tunnel-head; the ordinary products of combustion being wholly or partially absent.

In spite of these apparent advantages, this type of furnace has not been tried anywhere on a commercial scale, and the inventor himself, though retaining the principle of "column charging," prefers, in practice, to omit the central tube.

At present, therefore, Mr. Austin prefers to use a modified, and less complete, form of column charging, in which there is no absolute division of the ores, the siliceous and pyritous materials being only so far separated as may result from their different distribution in the furnace. The siliceous ores are fed close to the walls of the shaft, the pyrites being placed in the center.

I cannot understand how these two classes of material escape a more or less complete mixing long before they reach a point where any important reactions begin. Any one familiar with blast-furnace work knows well how rapidly the charge sinks in the center and lags behind on the sides, the ridge of outside material eventually curling over, like the crest of a breaker, and sliding toward the middle. This difference in the rate of sinking is still further intensified by the blast, which always hugs the walls, and thus tends to buoy up the light siliceous ores, while the heavy sulphides in the center, being largely in a fine condition, are impermeable.

Thus it will be seen that this method tends to shade imperceptibly into my second subdivision.

PYRITIC SMELTING WITH LAYER CHARGING.

Until the distinctive method of column charging is put into actual practice, so that positive results can be obtained as to its utility, it will be more convenient to treat of the entire practical side of the subject under the present section.

Pyritic smelting with layer charging is a compromise process. It seems to me that it has attained its present importance, because there is no prominent mining district in the United States where, with our present limited experience, the ores are exactly suited to the more radical and distinctive method of column charging.*

* It must be remembered that we dare not, without further experience, attempt to produce a matte by this highly oxidizing process, running high in copper, so that the process is, as yet, limited to districts containing siliceous silver or gold ores, with sufficient supplies of cheap, massive pyrites, high in iron and low in copper, to act as a fuel. In addition, the pyrites itself should carry a reasonable value in the precious metals, while all of the ores should run low in lead and zinc. Such a condition of affairs would be unique, and will, doubtless, not be found essential when the process has been more thoroughly ex-

Although the general reactions and end-results are pretty much the same in both column-charging and layer-charging methods, the means employed to arrive at these results are very different.

In column charging, we have:

The comparatively cold pyrites fed at once into the smelting zone, where an intense heat is produced by its sudden combustion.

A powerful hot blast, and an actual bessemerizing effect, independent of the shape of furnace, and positive in its action.

The privilege, and indeed necessity, of rapid driving and great capacity; and if we desire to avoid undue losses, the production of a slag approaching a bisilicate.

In layer charging, we have:

In order to obtain the best results, a set of conditions almost diametrically opposed to those just enumerated.

The pyritous, siliceous ore (and added coke) so fed, that they are intimately mixed and gradually heated in their descent through the furnace shaft.

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ploited. The cupriferous pyrrhotite of the Ely belt in Vermont, and the massive copper-bearing pyrites of the Verde mines in Arizona, possess an abundance of ores exactly adapted to a pyritic process, as well as ample siliceous ores for fluxing. But, apart from the local occurrence of rich precious metal ores at the Verde, these are strictly *copper* mines, where it would be necessary to make a 50 per cent. matte at the first smelting, and the question of losses in the slags at once arises. In Canada there are several important districts that would be greatly benefited by pyritic smelting. The Tilt Cove ores of Newfoundland (these are now being smelted by the pyritic method with reported success) and the extensive bodies of pyrites in the Crown and Albert mines at Sherbrooke, Quebec, would be admirably adapted to this method, if sufficient siliceous ores could be obtained to obviate too basic slags. But the most favorable conditions that I know of in North America are presented by the nickel-copper ores of Sudbury, Ontario. The ore is a nickeliferous pyrrhotite, containing some 8 or 10 per cent. of chalcopyrite, and with sufficient siliceous gangue (diorite) to form an excellent slag, which could be varied from 25 to 35 per cent. silica, as desired, by a proper mixing of the ores from the different stopes. The alkalies and alkaline earths of the diorite assist in forming a much better slag than can be obtained with only ferrous oxide for a base. As coke is also comparatively dear while wood is cheap, the conditions for genuine pyritic smelting seem about as favorable as could be desired; but, in common with all the localities mentioned, we are confronted with the possible danger of too heavy losses of metal in the slag, aggravated, perhaps, in this case, by the still greater oxidizability of the nickel.

If it were not for this unsettled point, the selected copper ores of the Spanish Rio Tinto mines, and above all, the enormous masses of cupriferous gold and silver-bearing pyrites of the Mount Lyell mine of Tasmania could be treated by this method with great economy.

A low blast pressure of considerable volume, by which the ore, during its descent, is slowly roasted, as in a kiln. While a hot blast is exceedingly useful, and certainly obviates a large proportion of the irregularities that so often occur, it is not a *sine qua non*, as in the former process. (The ore is not so thoroughly roasted but that it still contains nearly enough sulphide of iron to complete its own fusion by the time it has reached the zone of energetic oxidation, just above the tuyere level.)

No contraction of the furnace shaft toward the bottom.

A slow rate of smelting, to give time for the furnace to exert its kiln-like influence in roasting the ores, and thus also preheating them for the actual fusion at a lower level.

When all conditions are favorable, and the furnace is doing good work, the gases at the tunnel-head are no hotter than in ordinary copper-smelting. They are thoroughly oxidized, and thus absolutely useless for any further heating purposes.

If the slag is highly aluminous or siliceous, and the proper temperature cannot be maintained in the smelting zone without the use of too much coke (which counteracts the roasting influence, and produces low-grade matte), or without contracting the area of the furnace at the tuyeres (which at once also lowers the grade of the matte, as before explained at length), it seems to be judicious to narrow the entire shaft, still retaining the perpendicular walls.

As the blast pressure is light, and the rate of smelting purposely slow, there is only one way in which we can maintain a reasonable capacity for the furnace. This is by lengthening the shaft, and building the furnace in the shape of a long, narrow rectangle. Ores always roast best next to the walls of a smelting furnace, on account of the tendency of the blast to hug these surfaces, and the longer and narrower the shaft, the greater the wall surface in proportion to the area of the cross section.*

* This is exactly the opposite of the conditions that prevail in the stall and kiln-roasting of sulphide ores, where the roasted ore next the wall usually contains more sulphur than that nearer the center of the apparatus, owing to the cooling influence of the wall.

This conclusion was arrived at, apart from abundant ocular proof, by a considerable number of careful tests, among which the following are from large samples, selected by myself from stalls and kilns, where the roasting seemed as nearly perfect as is attainable under such conditions, and often with material that is not as free-burning as could be wished.

One set of samples was taken from a 6-inch layer of ore (as nearly as practicable) next the walls, while the other sample represents the entire remainder of the roast.

ordinary smelting, furnaces 11 and 12 feet in length by 3½ et in width are used to great advantage, and I can see no on to constructing pyritic furnaces for layer charging that e 32 to 36 inches in width by 180 inches in length, and with dicular walls. The height must be varied according to the rosity, and other physical qualities of the ores to be treated, ere should probably never be less than 4, or more than 7 an ore column. With a shallow column, it is sometimes ible to avoid overfire, or to utilize the heat of the escaping while too high a column of ore is apt to offer too great a ice to the blast, and does not permit a prompt modification composition of the slag.

irnace of the size mentioned, when kept free from wall ons, will have an effective smelting area of some 45 square more, and running on moderately favorable ores, and with olume, but low pressure of blast, should have a capacity of 150 tons per 24 hours.*

tic smelting with layer charging is probably as elastic a

OF COMPARATIVE CONTENTS OF SULPHUR IN ROASTED ORE, OF LAYER NEXT THE WALLS OF THE STALL OR KILN.

of Mine.	Treated at	Apparatus.	Per Cent. of S in 6-inch Layer, Next Wall.	Per Cent. of S in Remainder of Roast.
....	Butte, Mont.	Stall..........	13.4	8.25
Lyell....	Tasmania ...	Stall (exper'tal)	9.6	6.2
ill.......	East Boston	Stall, roofed...	6.2	4.1
....	New York..	Stall, roofed...	7.7	5.8
....	From acid kiln, N. Y.	Kiln..........	5.1	3.9
ita.......	Sonora......	Stall..........	8.96	6.2
te, 36 per	Ely, Vt.....	Stall, roofed...	13.3	9.9
te, 71 per	Ely, Vt.....	Stall, roofed...	11.1	8.6
rage ...	....	....	9.42	6.56

natte next the walls, therefore, contains an actual 2.86 per cent. more than that which was not so directly exposed to their cooling influence; ng the amount of sulphur in the middle ore 100 per cent., the outside tains about 44 per cent. more of this metalloid.

* indebted to Mr. F. A. Bartlett, of Cañon City, Colorado, for my first tic ideas on many points connected with this kind of smelting. He nicated them in several letters written to me about the year 1884, and e was managing the smelter at Portland, Maine.

process, and one adapted to as varying and refractory classes of ore, as any furnace method now practised.

The slags may range in composition between unusually wide limits.

he degree of concentration may be varied at will to an extent unattainable by any other furnace method. Injurious metalloids, such as arsenic, antimony, and tellurium may not only be gotten rid of more effectually than in either blast or reverberatory smelting but may actually furnish, by their combustion, a valuable amount of heat; while zinc and lead, though unwelcome constituents, produce no worse effects than in ordinary blast-furnace work, though they certainly cause more inconvenience and heavier losses of silver than in the reverberatory furnace.

The following table, showing the behavior of various substances in the atmosphere of the ordinary reducing blast-furnace (called in the table the German system), and in the oxidizing atmosphere of a pyritic furnace is taken from a most valuable paper by Herbert Lang, in which more insight is given into the true inwardness of pyritic smelting than in any other material that has been published.*

COMPARISON OF SMELTING EFFECTS.

SUBSTANCE.	GERMAN SYSTEM.	PYRITIC SYSTEM.
Quartz	Scorified	As in German system.
Alumina and its compounds	Scorified	As in German system.
Limestone	Co_2 volat CaO scorified	As in German system. As in German system.
Magnesian Limestone	Co_2 volat CaO, MgO scorified	As in German system. As in German system.
Lime, Magnesia, Baryta and their silicates	Scorified	As in German system.
Heavy Spar	So_2 (So_3 ?) volat. BaO scorified. BaS enters matte.	So_3 volat. (conjectural). BaO scorified. $BaSO_4$ fused, eliminated with basic slag.
Iron Pyrites (Pyrite) (Pyrrhotite)	S volat. FeS matted.	SO_2, SO_3 volat. FeS matted. FeO scorified.

* "Improvements in Matte Smelting," by Herbert Lang, ***Mining and Scientific Press***, January 12, 1895.

COMPARISON OF SMELTING EFFECTS—*Continued.*

SUBSTANCE.	GERMAN SYSTEM.	PYRITIC SYSTEM.
Iron-Copper Sulphides (Bornite) (Chalcopyrite)	(Cu Fe) S formed and matted.	(Cu Fe) S matted. FeO scorified.
Copper Carbonates and Oxides.	Copper enters matte as Cu_2S.	As in German system.
GOLD *in any form*......	Enters matte as sulphide; (?) recovery complete.	As in German system. Recovery complete.
SILVER *in any form*....	Enters matte as sulphide.	As in German system. Recovery probably decreases as pyritic effects increase in intensity.
ZINC-BLENDE	Part enters matte. Part decomposed— { ZnO enters slag. Zn volatilized. S volatilized. }	Largely decomposed. ZnO scorified. part volatilized. SO_2, SO_3 volatilized.
GALENA.	Enters matte as PbS or Pb_2S; recovery of lead complete.	Mainly decomposed. PbO scorified. SO_2, SO_3 volatilized. Recovery of lead incomplete.
ARSENIDES and SULPH-ARSENIDES (Mispickel, Lollingite, Leucopyrite, etc.).................	Arsenic slightly volatilized; remainder fuses with metals of group 4, as arsenide matte (speiss).	Arsenic chiefly volatilized. Iron oxidized and scorified.
COBALT and NICKEL....	Enter matte to the exclusion of iron, but are excluded by copper. In presence of arsenic enter speiss. Recovery complete.	As in German system.
METALLIC IRON........	Taken up by sulphur, forming matte. Scorification trifling.	Oxidation complete. Scorification of resulting FeO. Smelting temperature produced, and process assisted.
LEAD, *as Oxides and Carbonates*............ ..	Matte as sulphide. Recovery perfect.	Mainly scorified. Part may enter matte. Recovery imperfect.
MATTE (Fe, Cu, Ag, Pb, Co, Ni), (As, S).	Fuses unchanged.	Part oxidized. Separation of As as As_2O_3; S as SO_2; Fe as FeO; Pb as PbO; fusion of remainder as (Cu, Co, Ni, Au, Ag) S. Oxidation

COMPARISON OF SMELTING EFFECTS—*Continued.*

SUBSTANCE.	GERMAN SYSTEM.	PYRITIC SYSTEM.
		(and concentration of matte) proportional to intensity of pyritic effects. Losses of lead and silver probably dependent on composition of resulting second matte.
COKE................ CHARCOAL...........	Incomplete combustion. Products CO, CO_2 in various proportions.	Complete combustion. Product CO_2.
COAL..................	Distillation of volatile constituents. Incomplete combustion of fixed carbon. Calorific effect limited.	Combustion probably complete, and calorific effects satisfactory (conjectural).
WOOD..................	Distillation excessive. Very slight smelting effects.	Combustion tolerably complete. Temperature may be sufficient for smelting, especially if aided by the hot blast.

I speak especially of silver, for, from what has already been stated, it will be perfectly evident that pyritic smelting at its present stage of development, is a precious-metal, rather than a copper, process. It is, in fact, a method as yet mainly suited to the recovery of the gold and silver in ores containing a small percentage of copper; or, indeed, in ores that contain no copper at all, providing always that there are sufficient, though usually very minute, quantities of other elements present, to carry the values into the sulphide of iron, which here represents the matte.

The frequency of the association of gold, silver, and copper in ores warrants the consideration of this subject in a work on copper smelting.

In the precious-metal regions of the West, suitable pyritic ores are extraordinarily scarce. The silver and gold ores themselves are usually extremely siliceous, very low in copper, and comparatively high in zinc. They usually carry just enough lead to make trouble in pyritic smelting, but not sufficient to be suitable for a lead-process, while their values of gold and silver are not usually contained in the galena; hence, they cannot be concentrated without heavy losses. A large proportion of them are too low-grade to stand ordinary treatment, and yet occur in such abundance that they seek any reasonable outlet.

It is largely with this class of material that pyritic smelting has had to do. Hence, we have no records of long runs on proper material, on which to base our calculations of costs and results.

The scarcity of pyritic ores has nearly always forced the metallurgist to unduly increase his proportion of coke, and to employ barren limestone as a flux; and it has been sometimes hard to say whether he was practising pyritic smelting with an unusual proportion of coke, or whether he was doing ordinary coke-smelting, and tryin to get some little help from the oxidation of the trifling amounts of sulphur and iron in the ores.

In pyritic smelting with layer charging, a certain amount of carbonaceous fuel (coke, charcoal, wood, or coal), seems essential, if only for its mechanical effect in loosening up a dense charge.

The only reliable results, therefore, that can be furnished, are exceedingly unsatisfactory, and totally unfair to the method, the conditions in no single instance being favorable. But, as it is important to metallurgists to know exactly what has been accomplished in this field, I publish the results of the most important pyritic runs, as they are interesting in showing the capabilities of this method, even under very adverse conditions.*

The use of the hot blast is not so obligatory in this method of pyritic smelting as it is where column charging is practised; yet it is of such great utility in correcting variations of temperature and other irregularities, that I can hardly conceive conditions, in genuine pyritic smelting, under which it would be more profitable to omit it; and where the ores are at all refractory, and a highly siliceous slag must be produced, it becomes practically a necessity.

SUMMARY FOR PYRITIC SMELTING.

1. Genuine Pyritic Smelting (without the addition of any carbonaceous fuel) demands exceptionally favorable ores, and has not yet been brought to a sufficient state of perfection to warrant its installation otherwise than experimentally.

2. Partial Pyritic Smelting, with the aid of a hot blast, and the addition of 1.5 per cent. to 5 per cent. coke to the charge, is an assured success, and may, under proper conditions, be the most economical method that can be employed.

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* See chapter XV., by Mr. Robert Sticht.

3. Until we acquire more experience as to the loss of copper in the slags by scorification, it will not be safe to employ pyritic smelting as a straight copper process, or where a first matte of over 20 per cent. to 25 per cent. copper is to be produced.

4. Both pyrrhotite and matte make excellent materials for this process, pyrrhotite affording practically about as much heat as does iron pyrites, owing to the loss of the first atom of sulphur in the latter mineral, by volatilization as metallic sulphur. The oxidation of the iron probably furnishes as much effective heat as the burning of the remaining atom of sulphur.

5. Arsenic and antimony are more thoroughly removed and thus cause less trouble than in ordinary smelting. Zinc and lead sulphides are about equally deleterious in the new, as in the old, method of smelting. Heavy spar works peculiarly well in a pyritic furnace, the baryta being slagged and the sulphuric acid decomposed and escaping as gas, instead of augmenting the quantity of matte, as in ordinary blast-furnace work.

6. The recovery of the precious metals is as good in pyritic, as in ordinary smelting, and the slags equally clean.

7. The rate of concentration is very satisfactory under proper conditions, and can, in the main, be regulated at will, according to shape of furnace, volume and pressure of blast, fusibility of charge and rapidity of driving, and amount of carbonaceous fuel added.

8. A plant for pyritic smelting can be erected at least as cheaply as one to produce similar results by the ordinary methods.

If it were subsequently decided to change back to ordinary smelting, the plant would simply require the addition of calcining furnaces and crushing facilities therefor, and I am inclined to believe that the costly stove for heating the blast would be a most valuable adjunct to our ordinary blast-furnace work. It would enable the smelter to produce cleaner and more siliceous slags (the latter, a great object at many localities), and I do not think would cause the irregularities and the reduction of iron that were found to occur when it was applied to the small, slow, internal-crucible furnaces of a former period.*

* The hot-blast is used to a limited extent in various copper furnaces in Norway and Germany; unfortunately, I am unable to obtain details of the results in time for publication in this edition. But it is idle for writers to keep quoting the Mansfeld practice as an example of the successful application of the hot-blast to ordinary copper smelting. The Mansfeld conditions are unique, the

9. It is a sheer waste of time and money to attempt the installation of a new pyritic plant without taking advantage of the experience on the subject already so dearly accumulated.

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ore being a slate, almost barren of metallic gangue, and no fluxes being used. The slags are bi- and tri-silicates of alumina and lime, and carry only 3 per cent. to 7 per cent. iron. Even with their comparatively rapid smelting, iron is reduced as sows, causing a very disagreeable and expensive periodical clearing-out of the crucible. These sows are sold in England for the price of iron, to be there worked up for the small quantities of nickel and cobalt which has become concentrated in them.

CHAPTER XV.

PYYITIC SMELTING—ITS HISTORY, PRINCIPLES, SCOPE, APPARATUS, AND PRACTICAL RESULTS.*

DEFINITION OF PYRITIC SMELTING.

DR. JOHN PERCY'S "*Metallurgy*," *Silver and Gold*, Part I., p. 531, defines "Pyritic Smelting" as follows:

"By Pyritic Silver Smelting is meant the smelting of silver ores, which are either free from lead, or do not contain this metal in sufficient quantities to collect the silver, in conjunction with pyrites, in order to produce a regulus in which the silver shall be concentrated. Iron pyrites is used for this purpose, and by preference, such as is argentiferous or auriferous; but failing this, cupriferous iron pyrites, or copper pyrites, may be substituted."

The practice at Sala in Sweden and Kongsberg in Norway is given as an example; a prominent feature of the method being, that by means of pyrites it is possible in a single smelting operation to get rid of the great mass of foreign matter in the ore, by converting it into a worthless slag, and thereby to obtain the silver in a concentrated shape in the regulus.

This concentration not being high enough, it is further stated, that by partially roasting this regulus and again smelting it with the addition of a due proportion of silica, a second regulus, much richer than the first, may be obtained; or, if siliceous ores be used instead of straight silica, the concentrated regulus may be made still richer in the valuable metals.

A study of the details will show that the first smelting for regulus, as well as all subsequent concentration of the same after pre-

* The great importance of the principles involved in "Pyritic Smelting," and the interest manifested in this process by copper metallurgists throughout the world, has induced me to apply to one of the few men practically experienced in this branch for assistance in my treatment of the subject. My valued friend, Robert Sticht, of Montana, has kindly prepared the following chapter on Pyritic Smelting, and I take pleasure in publishing it at length.

paratory roasting, is effected with a liberal use of carbonaceous fuel. The amount of this fuel at Kongsberg, for instance, reached 22 per cent. of the weight of the charge, or some 40 per cent. of the weight of the ore treated, the charge consisting of nearly equal amounts of ore, and slag from other operations. (Attention may also be called to the fact that a hot blast has there been used on the blast-furnaces for more than forty years past.)

It will also be noticed that the proportion of raw iron pyrites thus used is comparatively small, being at Kongsberg but 15 per cent. of the charge. Furthermore, it is evident, that the main role of this pyritic material here is but to act as a collecting agent of the precious metals present (and the copper), and that this "Pyritic Smelting" has not for its object the treatment chiefly of excessively pyritic ores, such as those of Rio Tinto, or Mt. Lyell, for example. There is in it nothing more required of the sulphur contained in the pyrites than the assertion of its affinity for the gold, silver, and copper contained in the ores, together with some of the iron present. No importance at all is attached to the calorific value of the sulphur; that is, to its property of being itself a metallurgical fuel. Nor is any such demand made on the similar attribute of the iron associated with the sulphur.

Neither in the ore nor in the regulus is either of these substances regarded as a possible source of high heat, and utilized accordingly. On the contrary, in the treatment of the regulus, the sulphur is very carefully nearly all removed by roasting, with the aid of more or less carbonaceous fuel, and this desulphurized product is then smelted with the aid of still more extraneous carbon.

The scheme thus outlined in the case of the regulus, in the process of this older "Pyritic Smelting," is also the one which is universally followed in the smelting of sulphureted copper ores; that is, ores in which the value of the copper approaches, or exceeds, that of the precious metals. It is the method which established custom would pursue even in the case of such highly pyritous ores as the two called "excessively pyritic" above, just as it is now everywhere followed in the case of ordinary ores, carrying larger proportions of earthy gangue than these.

Whenever the latter constituent greatly predominates, as in the ores of Butte, Montana, a preliminary wet concentration is resorted to, which is simply mechanical, for the purpose of making the metallurgical treatment a profitable operation. But in all cases, the ordinary method of the day is, first, to drive off nearly all the sulphur, with a total disregard of its capacity to provide heat itself sufficient for fusion, retaining only that portion of the sulphur that is needed for the purpose of holding together, in the shape of matte or regulus, the gold, silver, and copper; and then to have recourse to a foreign element, carbon, in some one of its various forms (coal, wood, coke, or gaseous combinations), for the heat for fusion to matte.

Now, in distinction from the meaning of the term "Pyritic Smelting," as applied to the initial use of raw pyrites as a mere collector of values, as well as in distinction from the ordinary methods in which no fusion is done without a previous roasting, I wish to lay stress on a recent introduction into the field of metallurgy, or rather, the modern revival of an idea fallen into desuetude about fifteen years ago, which is peculiarly adapted to the treatment of fairly, and excessively, pyritic ores. This method assigns such an important part to the pyritic nature of the ore itself, that the name "Pyritic Smelting" ought really to be restricted to it. It is a process full of the greatest promise for the future.

The process may be described as the smelting of raw, unprepared, pyritic ores, or similar natural or artificial sulphides, direct in the blast-furnace, by means of the heat generated from the rapid oxidation of certain constituents of the ores or substances themselves, and with the addition, if necessary, of siliceous or calcareous fluxes only. No carbonaceous fuel, or, at most, only a slight proportion of the same, is used with the ores.

No preliminary chemical preparation of the material, such as roasting, is required. Therefore, the various sources of outlay for this laborious and sluggish operation, such as apparatus, fuel, labor, repairs, and the tying up of unused capital for long periods of time, are entirely done away with.

Since constituents of the ores themselves are made to furnish the requisite amount of heat, the very heavy item of expense for coke or charcoal used in the blast-furnace is escaped, except for a very small, hitherto unavoidable proportion.

It must, however, be remarked that the great advantages to be derived from the use of a heated blast have, up to the present time, enforced the employment of the same; thus there is an expense for cheap, carbonaceous fuel for this purpose. But this is used outside of the blast-furnace, and the expense is slight.

In brief, the object of pyritic smelting proper is to make the pyritic, or other sulphureted material of the ores, act in a unique and threefold capacity, viz.:

First. As a fuel.

Second. As a collector of the precious metals, as well as of the copper, nickel, etc., present.

Third. As a flux, or slag-making factor.

The first and third peculiarities distinguish it from Dr. Percy's pyritic smelting.

The first peculiarity is the chief point in which it differs from the ordinary smelting of sulphide ores of copper.

Compared with pyritic smelting in the older sense, the process is applicable to much more highly ferruginous ores, or rather to the working up of a greater proportion of such.

The first feature, it is also proper to add, is the one which restricts the process to such ores only as have a sufficiency of fuel in them (*i.e.*, sulphur and iron), or such mixtures of ores, etc.

However, given an ore which is favorable, it is surely not progressive, nor economically rational, to adhere to the established method of first roasting, then smelting, and, perhaps, subsequently re-roasting and re-smelting, etc., when both time and labor can be reduced and money saved by avoiding the costliest items of expense for materials and handling nearly completely, by a proper utilization of the constituents of the ore as given by Nature; and this without any greater smelting losses than are inevitable when employing the most perfect modern processes.

This method is applicable to the greatest variety of pyritic ores; argentiferous, auriferous, and cupriferous. That is to say—to straight iron pyrites and straight copper pyrites, and to mixtures of the same in any proportions. All natural or artificial metallic sulphides and sulpho-arsenides, with non-volatile, metallic bases are suitable. Mattes and speiss may be concentrated by it, either alone, or in conjunction with other ores. Magnetic pyrites (pyrrhotite) is an excellent material for it. The whole line of sulphureted copper ores is favorable. Nickel and cobalt ores are not excluded, and oxidized ores of any kind can be treated along with the sulphides.

Though quite as sensitive to certain substances, like the sulphides of lead or zinc, as ordinary matte-smelting, it is less so to others, as arsenic, antimony, and sulphate of baryta.

Hollway retains a bath of molten matte in the furnace & f[orces]
a blast of air or other oxidizing gas through the molten m[atte]
- depresses the tuyeres to do so

HISTORY OF PYRITIC SMELTING.

The idea of employing the compounds of sulphide substances as sources of great heat originated with a Russian engineer, Semennikow, about 1866, whom the example of the steel Bessemer process, which dates back in its first attempts to 1855, induced to recommend the application of the same fundamental principle to copper mattes. Thus Semennikow gave the first impulse to the process now known as Manhès', and successfully practised for the past fifteen years.

Although the employment of the same method for natural sulphide ores seems a very near-lying thought, there is no record of practical research in that line until the time of Mr. John Hollway's extraordinary and successful experiments with Rio Tinto ores in 1878. To John Hollway of England, is due, therefore, the credit of being the founder of the smelting process advocated on these lines.*

The products were to be an enriched copper matte, or regulus, containing the gold, silver, and copper of the original ore; sublimates, containing part of the sulphur in its metallic state, and the lead, zinc, and rarer volatile metals; and last, but very important to Mr. Hollway and his associates, sulphuric acid, made from the sulphurous gases generated in the operation. For well-known reasons, our modern pyritic smelting has hitherto neglected to look upon the sublimated sulphur and the sulphurous acid gases as sources of revenue, although, technically, there is no insurmountable difficulty in the way of utilizing them. Again, metallic sublimates are not to-day supposed to be made in pyritic smelting, any more than in lead or copper smelting. The only product intended to carry metal values is the regulus (matte).

Mr. Hollway's essay, and its discussion by the Society, contain

* A very full account of the chemical principles involved, together with the preliminary theoretical calculations and detailed descriptions of the experiments substantiating and demonstrating the feasibility of the idea of using the sulphur and iron of pyrite for the purposes of fuel, may be found in an admirable essay of Mr. Hollway's read before the Society of Arts on the 12th of February, 1879. This essay is entitled, "A New Application of Bessemer's Process of Rapid Oxidation, by which Sulphides are Utilized for Fuel." It also contains an exhaustive discussion of the subject by the Society, and the estimates and designs for the treatment of 300,000 tons of cupreous pyrites per annum at Rio Tinto, the well-known deposits of that district having suggested the practical execution of the idea.

Depressed tuyeres make bottom of furnace into a sort of
Bessemer converter, in which the matte formed in
part of furnace is further oxidized, thus furnish[ing]
more heat to conduct the smelting process & at th[e]

tion – "subjecting the charge to the action of a hot blast and continuously drawing the molten pr
o formed away from the path of the blast"

so many foreshadowings of our present method of pyritic smelting, and its chemical principles are so well stated, that it is worth quoting at some length for the sake of a clear and precise enunciation of the primary facts of pyritic smelting proper. It will be the best exposé that could be given of the process which I am describing, to characterize it in words spoken when our present specific, practical realization of the idea was yet in embryo; because, as far as the scope and nature of the operation are concerned, these are already recognized and laid down in their full scientific truth. Up to date, our more recent experience has added nothing of importance to either modify or amplify the main points made by Mr. Hollway fifteen years ago.

He says: "When metals are extracted from their ores by fusion, the necessary heat is always obtained by the burning of coal, coke, or other form of carbon. I wish, however, to remind you that sulphides can be made to burn in air, and are thus combustible substances, while the oxides are bodies that have been already burned, which, as you know, is the conventional expression for entering into combination with oxygen. The metallic sulphides, consequently, are natural combustible minerals, and my object is to prove that they can be utilized as sources of heat in certain metallurgical operations. The most important of the mineral sulphides is pyrites, both on account of its occurrence and the extent of its deposits. The predominating constituent in this mineral species is bisulphide of iron, with which are frequently associated sulphides of copper and arsenic. Silver and gold are also usually present in larger or smaller quantities. When iron pyrites is roasted in the open air, an increase of temperature takes place in its mass, so that the oxidation continues without the application of external heat, the sulphur passing off as sulphurous acid gas, while the iron is changed into ferric oxide.

"This process of roasting extends over a considerable space of time, and is so conducted that the heat evolved by the oxidation of the sulphides is never very manifest at any period of the operation. The sulphur and metals frequently burn to fusion, but the utilization of the heat evolved by this burning has not hitherto been considered a subject of much importance. If, however, a rapid current of air is forced through molten sulphides, the maximum temperature of the combustion is attained, because all the oxygen of the air is then utilized for oxidation, and the oxidation is concentrated into the space of a few minutes instead of occupy-

-ned furnaces are advantageous in matte concen-
-tion – and maybe as good as horizontal in
ordinary pyritic smelting

ing many weeks, or, in the case of cupreous pyrites, several months."

This, be it said, is the salient feature of pyritic smelting proper, viz., the restriction of the oxidation of the sulphides to a small modicum of space and time, so that the heat generated by the operation is intensely concentrated, and achieves the specific object intended, namely, the fusion of the solid products of the operation. The great excess of air that characterizes roasting is also avoided, and thus, furthermore, the gaseous products have no opportunity to carry off an influential amount of the heat.

"Iron pyrites (FeS_2) upon fusion yields about one equivalent of free sulphur in the shape of vapor, and becomes protosulphide of iron, which rapidly absorbs oxygen when exposed to the air, the iron becoming a protoxide in a very short time. When air is forced through liquid sulphides, a very energetic action takes place, and the protosulphide becomes converted into protoxide with great rapidity. The reaction representing this decomposition is

$$FeS + O_3 = FeO + SO_2.$$

In practice, fluxes are present during this reaction, because the protoxide of iron requires silica to form a slag; and these fluxes can be introduced with the charge of pyrites. Such poor metalliferous substances as the gangue of mines, ancient scoriæ, poor, siliceous copper ores, etc., containing small quantities of valuable metals, may thus be advantageously utilized, and the copper, lead, and silver that they contain will be found in the regulus or sublimates. These materials can be introduced in considerable quantities, because their specific heat is low."

Needless to say, these substances need not necessarily be "poor," but any siliceous silver or gold-bearing material, with or without copper, may be used. It is absolutely necessary to have a certain amount of silica, for the purpose of slagging the protoxide of iron above mentioned, so as to allow of a clean separation of the matte and slag.

In this connection, it is very interesting to note the advantage inherent in pyritic smelting over ordinary smelting for matte. In the latter, the sulphide of iron is first roasted to ferric oxide (Fe_2O_3), or to peroxide (Fe_3O_4), in which condition it is not fit to enter into combination with silica, as this substance requires ferrous oxide (protoxide of iron, FeO). One function of the carbonaceous fuel required in ordinary smelting, is to reduce the over-oxidized iron back to the ferrous oxide stage, thus causing a relatively greater obligatory application of carbon than for mere fusion. In pyritic smelting, the oxidation of the sulphide stops of itself at the correct ferrous oxide point, and thus an important economic advantage is established in its favor.

Mr. Hollway's essay incudes thermo-chemical calculations verifying the expectation that the heat generated by his method is sufficient to maintain the solid products of combustion, *i.e.*, the matte and slag, in a molten condition.

This, of course, is the most vital part of the entire question, as otherwise the process would not be a feasible *smelting* operation. It will suffice for present purposes to give merely the results of these calculations.

Thus, the temperature produced by the combustion of the protosulphide of iron *per se*, is found to be 2,225 degrees C.; by the combustion of sulphide of zinc, 1,992 degrees C.; of sulphide of lead, 1,863 degrees C.; the combustion in each case being in air (the nitrogen of which has a depressing effect on the thermal energy manifested).

For the sake of comparison, the case of a coal is cited, containing 80 per cent. carbon, 5 per cent. hydrogen, and 15 per cent. non-combustible matter, the temperature derived therefrom being 2,787 degrees C., for complete oxidation (to carbonic acid). An example of the bessemerizing of pig iron gives a temperature of 2,748 degrees C.

All these figures would be somewhat modified by our present more exact knowledge of the subject, but are sufficiently correct to serve the present object.

As the melting point of regulus may be placed at 1,000 degrees C., it will be seen that pyritic smelting, correctly conducted, may be made a very successful operation. Preliminary calculation furthermore demonstrates that the utilization of the sulphur in sulphides as a fuel for metallurgical purposes, compares very favorably with other fuels in nature.

Mr. Hollway also gives results theoretically obtainable in the case of a specific working charge, applied to pyrites containing 90 per cent. FeS_2, *i.e.*, 48 per cent. iron, and 10 per cent. foreign matter, under the condition of an assumed necessary temperature of the operation of 1,000 degrees C., the reaction, moreover, to be as follows:

$$5\ FeS_2 + 12O = 5S + 4\ SO_2 + 4\ FeO + FeS.$$

The very important fact is demonstrated, that with air at the ordinary temperature, there is still sufficient margin of heat to allow of adding 20 cwt. of slag-producing, metalliferous substances to each ton of his pyrites without lowering the temperature of the reaction below 1,000 degrees C.

Prof. Richard Ackerman of Stockholm corroborates these inferences, and computes the temperature of the above reaction, with only the foreign matter of the pyrites present, at 2,179 degrees C., thus lending the great weight of his endorsement to Mr. Hollway's own *a priori* theoretical deductions.

However, all these mathematical proofs derived from the theory of heat, were merely introductory to the actual, practical demonstrations themselves.

These were very extensive, are detailed at length by Mr. Hollway, and were pursued with the scientific thoroughness of the celebrated investigators before, and by whom the trials were made. The results were absolutely confirmatory of the *a priori* calculations, and aroused a great deal of enthusiasm and hope among the witnesses at the time, as disclosed in the discussion accompanying the essay.

The trials were all made in the regular steel Bessemer converter at Penistone, and while proving the complete feasibility of the process, demonstrated that the converter was an awkward apparatus therefor. For it was soon observed that the operation was absolutely self-supporting, and that, consequently, a vessel which compelled intermittent work because it could not be discharged except by pouring out the entire contents, was not fitted for it. Hence, Mr. Hollway says:

"These experiments were carried out with an ordinary Bessemer converter; but for many reasons a Bessemer converter is not suitable for the reduction of cold pyrites to regulus, nor has it any arrangements for allowing the regulus to accumulate out of reach of the blast."

Again, in a passage which I will quote at length on account of its great historical interest, and as foreshadowing the salient features of our present pyritic practice, he strikes the keynote of all recent permanent improvements and success, by saying (1879):

"It is probable that the form of furnace eventually adopted will be a modification of the ordinary blast-furnace, fitted with a tuyere-hearth. Such a furnace, built on pillars, with boshes and hearth of some substance not acted on too rapidly by the slag formed during the burning of the sulphides, would, working continuously, treat a large quantity of material. Being built on pillars, the crucible, hearth and tuyere-bottom could be replaced when necessary, without disturbing the remainder of the structure; and, as these would be the only parts in contact with the fused material, the furnace, from the boshes upward, should not experience much wear and tear.

"When a gannister lining, similar to the ordinary Bessemer lining, is employed for the boshes and hearth, the corrosive action of the protoxide of iron would be neutralized and avoided by introducing, with the pyrites, sufficient siliceous material to produce a slag containing at least as large a proportion of silica compared with the bases, as the formula 2 RO, SiO_2. If, however, a basic lining is employed, the slag should contain less silica, and in no case more than the proportion expressed by the formula just given. Under such circumstances, the blowing would be continuous, the hot charge coming to a fusion zone, the height of which over the tuyeres would be determined by the amount of air blown in, and the frequency with which the blown products are withdrawn, varying likewise with the composition of the charge.

"The products would be withdrawn by tapping, as with a common blast-furnace, the regulus being run off from a reservoir below the tuyeres, where it would collect, and being thus unacted upon and undisturbed by the blast, rich regulus, or even metallic copper, could be produced. By continuing the oxidation, and producing Cu_2S and some metallic copper (smelting for bottoms), the gold and silver would be found with the metallic copper.

"A large side-flue at the top of the furnace would carry off the gases after their temperature had been reduced in heating the charge introduced above through a self-closing hopper. It is calculated that such a furnace, 30 or 40 feet high, with a hearth capacity of one cubic meter, would be capable of treating annually 50,000 tons of pyrites and a similar quantity of siliceous fluxes, working 200 days per year.

"The theory of melting sulphides with a blast-furnace is as follows: The operation is started by placing the tuyere-hearth in its place and throwing in hot coke at the top of the furnace. The blast is now turned on, and the coke develops a high temperature by its rapid combustion. The ordinary working-charge of sulphides and fluxes is now introduced at the top hopper, and as the sulphides melt, the coke burns away. As soon as a layer of molten sulphide lies over the tuyeres, the blast is increased, and also the burden of ore. The charge above the fusion zone is gradually heated as it descends, losing much of its sulphur by volatilization before it becomes molten. On fusion, considerable lead sulphide will distil over, accompanied by the remainder of the arsenic as sulphide, in the strong current of nitrogen and sulphurous acid. These gases, as they pass upward in the furnace, will be greatly reduced in temperature by the volatilization of the sulphur and moisture from the crude materials. There is reason to believe that more than half the sulphur in iron pyrites is volatilized in the free state by this operation. The sublimed sulphur, sulphides, and oxides would be collected in the wide chambers with which the side-flue is connected.

"Below, in the hearth, the oxygen of the air forced in acts upon the sulphides of iron and zinc contained in the charge, and as long as a constant supply of these substances arrives at the hearth, no other constituents present will be appreciably oxidized. A tap-hole near the top of the hearth will allow the slag to be withdrawn. The blowing would be continuous day and night, as long as the tuyere hearth lasted, and the heat from the gases, as they leave such a furnace, could be utilized so as to heat the blast, or produce steam-power for the blowing engines. If desired, the products could be run direct into suitable reverberatory furnaces, where, after the regulus had subsided, the slag would be run off while yet in a molten state, and in which the oxidation of the regulus could be completed."

Thus says Mr. Hollway, in elaboration of convictions forced upon him by the results of his Penistone experiments.

Excepting for allowances that have to be made for the fact that the subject is regarded in these words from the point of view of an iron-master, *i.e.*, great size and capacity of furnace, etc., the above quotation is, in the main, a concise description of our present system of pyritic smelting. Some expectations of Mr. Hollway's are not borne out by fact, and some things have since been achieved or solved in a slightly deviating fashion, but the chief items of his prognostication have come exactly true. And this almost by an accident, as far as these items are concerned. It may be said that, as concerns the *letter* of above quotation, the simple circumstance that the problem passed out of the magnificent compass of activity to which the iron industry is accustomed, to the smaller and narrower lines of the metallurgy of the more precious metals, served to avert the difficulties inherent in it, and to rescue the principles of the process out of the false light of impracticability into which it subsequently fell.

There are certain mechanical difficulties, due to the nature of the ores treated, which do not allow of the use of the stupendous machinery employed in iron smelting. The substitution of our own smaller furnaces facilitated the surmounting of these obstacles, accidentally. Incidentally, as the modern water-jacket furnace for copper smelting was, of course, the type employed, it also put aside the whole interdependent questions of slags and furnace linings.

The direct production of metallic copper in the blast-furnace itself by pyritic smelting, as suggested in the first paragraph quoted, is made unadvisable by practical considerations. It has never yet been attempted, though patents have been issued on the idea. Neither has the "suitable reverberatory furnace" mentioned at the close, into which it proposed to tap the entire body of molten products, for the purpose of separation and discarding the slag, while the matte is blown to blister copper, materialized.

Reverberatory forehearths, indeed, are in use here and there, though not needed where the blast-furnace practice is good; and the Manhès process of treating the matte in a vessel separate from the forehearth will, for practical reasons, probably hold its own for a long time to come. However, the general reasoning is the same as that at the foundation of our present methods. We can now obtain a final product containing but a trifle less than 100 per cent. of all the copper, gold and silver contained in the ore, out of the latter, in the short period of about six hours from the moment of feeding the raw ore into the blast-furnace, intermediate delays between the latter and the Manhès converter not being taken into account, of course. In noteworthy contrast to the text of the essay, in the matter of the size of the blast-furnace suitable to the process, the plans designed for the treatment of "300,000 tons annually of cupreous pyrites" contain an explicit drawing of a "converter" to be used in this hypothetical plant (for Rio Tinto) as the most essential apparatus, which is nothing but a veritable blast-furnace of small dimensions—the entire efficient height from the shaft up being about six feet, and the internal diameter three feet. (See Fig. 51.) There is no allusion to this construction in the text, though it is actually far better adapted to the requirements than a furnace 30 or 40 feet high. This converter combines features of the stationary (non-tilting) Swedish steel "converter," which suggested it, with those of a blast-furnace. It differs from both in being mounted on wheels (like the modern Manhès converters of southwest Europe) so that it may be removed and

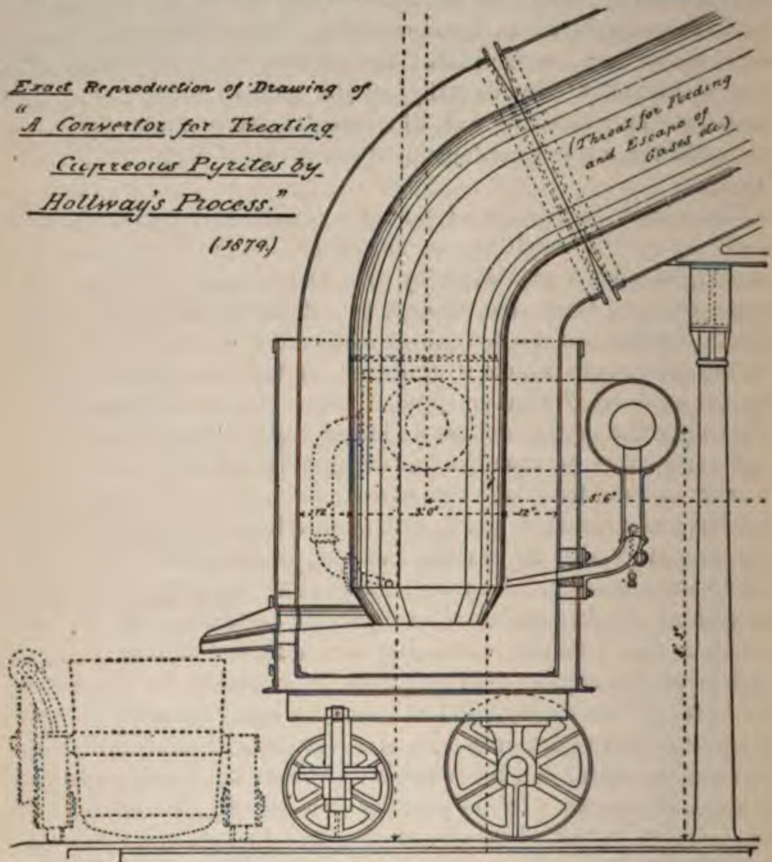


FIG. 51.

another substituted against a permanent system of condensation chambers designed to catch the sublimates. It has the horizontal floor and tap-outlet of a blast-furnace, and there are only three tuyeres placed about 9 inches above the floor, instead of on the level of the same, as in the Swedish converter, thus presenting more essential peculiarities of the blast-furnace than of the stationary converter. The tuyere connections are the ordinary ones of blast-furnaces, and fixed in place, but not permanently, to the "converter," if the drawing is not misleading, then periphery and bottom of apparatus were also designed to be cooled by water, but there is the usual gannister lining inside of this water-jacket. Thus, altogether, except for this lining, there is considerable identity with a diminutive copper-furnace of the present day. The furnace was probably never built and used. It would have done fair work on a limited scale, with a hot blast and low pressure.

Mr. Hollway's résumé of the principles which he logically demonstrated, and which are the bases of our own pyritic smelting proper, is as follows:

First. That the whole of the oxygen of the air driven into a thin stratum of protosulphide of iron (FeS) is utilized for oxidation.

Second. That by the heat involved in the rapid oxidation of sulphides, and without the use of extraneous fuel other than that employed in producing the blast:

(*a*) About one-half of the sulphur contained in iron pyrites (FeS_2) is expelled in the free state.

(*b*) The remainder of the sulphur, excepting that left with the regulus, is principally evolved as sulphurous acid.

. . . .

(*d*) The volatile metallic sulphides, such as arsenic sulphide and lead sulphide are distilled off with sulphur.

(*e*) Iron being more oxidizable than copper, silver, gold, nickel, and certain other metals, these latter will be all concentrated in the regulus, provided an excess of sulphide of iron is always present.

(*f*) The protoxide of iron thus formed is converted into slag by the addition of the silica introduced with the pyrites.

(*g*) The more perfect fusion of the slag thus obtained prevents loss of copper by entanglement with imperfectly fused material.

(*h*) About 16 cwt. to 20 cwt. of incombustible material, having a specific heat of 0.15 to 0.25, can be added, per ton of pyrites, when a cold blast is employed, assuming that 1,000 degrees C. is the temperature necessary for the operation.

(*i*) The quantity of similar incombustible material can be increased to from 30 to 34 cwt. to each ton of pyrites operated on, when a hot blast of 500 degrees C. is employed, assuming that 1,000 degrees C. is the temperature necessary for the operation.

(*j*) Such incombustible material may contain larger or smaller quantities of valuable metals as oxides, which will pass into the regulus after double decomposition with protosulphide of iron present in the molten bath. Thus, silicates of nickel or copper would be converted into sulphides of nickel or copper, and be concentrated in the regulus.

. . . .

Seventh. That the new process could be advantageously employed by the copper smelters for treating rich copper ores on account of the great economy there would be in labor and fuel.

Eighth. That the cost of plant is small, compared not only with the quantity of material it would treat, but also on account of the additional profits derived from the new process.

What is here omitted refers to the furnace-lining and the blast, and also to the sublimates and gases, which, it will be remembered, played an important part in the commercial incentive to apply the process to Rio Tinto ores.

In conclusion, it must again be said that our recent experience has added nothing of vital importance to the fundamental facts and principles quoted above, as written fifteen years ago, and forgotten or neglected for a decade. It has merely elaborated the practical part of the chemistry and mechanics involved.

It will suffice to give only a rapid survey of the later history of pyritic smelting proper, since the abandonment of the process by the contemporaries of the *Penistone* converter-experiments, up to the revival of the idea in the United States.

In 1881, three years after the above, there was erected, under F. W. Barnes, at Majdaupek in Servia, a Bessemer plant and other furnaces for the Hollway Bros. sulphide process; the information at command concerning the fate of the process here is too meager and indefinite to be given. More accurate inquiries are pending. Tessier du Motthay's apparatus for the same purpose, said to have been used with success at Coumiens in France, probably is of about this same date.

Experimental tests seem to have been made quietly at various places in the early eighties; very likely all in converters, and apparently all failures, to the detriment of the reputation of the process, the feasibility of which in the course of time, came to be referred to as "an iridescent dream." I know of one case in the United States where failure was due to the giving way of the bronze bottom-tuyeres employed, and the question of expense prevented renewed trials.

The so-called Manhès process for treating copper and nickel mattes, which is based on identically the same principles, may be said to have been a commercial success since 1883.

Very recently (1893), J. B. Hannay in England has treated lead sulphide ores similarly, while B. Roesing in Germany (1892), has even successfully applied the Bessemer principle of utilizing latent heat to the refining of lead bullion (treatment of zinc-crusts and cupellation of bullion in converters).

However, Mr. Hollway's explicit proposal to "bessemerize" sulphide ores in the blast-furnace instead of in converters, was not put into continued actual practice anywhere until 1887–8 to 1891. It was then that a combination of local circumstances suggested to members of the Toston Smelting Company, an English syndicate doing business at Toston, Montana, U. S., the advisability of again attempting it in improvement of their barely remunerative practice of ordinary matte-smelting, which they were conducting with extremely pyritous ores—first roasting these and then smelting them. The suggestion was derived, more or less directly, from Mr. Hollway's pamphlet above quoted. As the apparatus at hand for the purpose at Toston consisted only of a low matting blast-furnace of small cross-section and the ordinary blowing apparatus suitable to such work, the mechanical difficulties at first encountered (due to the ready friability and decrepitating tendency of the local ores, as well as the agglutinating effect of the volatile sulphur and the low specific heat of iron-matte), were very considerable. The attempts were never systematically conducted, but the results were sufficiently good to counsel persistence. The ordinary cold blast was at first employed, but the introduction of a small iron-pipe stove for the preheating of the blast, as suggested by a knowledge of the practice at various European matting-establishments, made the management of the process easier. Subsequently, the employment of a considerably larger lead-furnace of the ordinary American type, which was altered to suit the new purpose, and the installation of a more adequate stove, improved matters still more. The alteration of the blast-furnace consisted in replacing the lead-crucible by a flat bottom-plate (drop-bottom style), with a floor of brick and tamping upon it. This plate rested on short columns and also carried the batteries of water-jackets. There were two of these batteries.

The former lead-furnace jackets were retained, and raised sufficiently to allow of introducing, below them, a series of hearth-jackets, also provided with tuyeres. Thus the furnace was provided with two sets of tuyeres, one above the other. The blast-connections, etc., were altered to suit the use of a hot blast. The object of two rows of tuyeres (vertically 12 inches apart) was, either to oxidize the volatile atom of sulphur for heat, or to concentrate, by means of the lower row, the matte made in front of the upper; or, probably, both purposes. As a matter of fact, however, the matte made in front of the upper row immediately ran into the lower (only the ordinary lead-furnace blast-pressures being available), and smelting was then confined to the upper tuyere level. Hence, the upper set was plugged and the furnace is now virtually nothing but an ordinary matte blast-furnace, with paraphernalia adapted to the use of hot blast. A Herreshoff matting-furnace was transiently employed, but discarded; the skill to manage this perfect apparatus probably not being vested in those in charge. A receiver, or forehearth of some kind, was used from the beginning for the separation of the matte and slag outside of the furnaces, and the "Spur-Ofen" principle of furnace discharge was also adhered to from the start, *i.e.*, the blast was not trapped, and no layer of matte constantly held in the furnace.

In February, 1889, Mr. Lawrence Austin, one of the organizers of the Toston Smelting Company, and metallurgist of the company, under whose direction all the trials were made, applied for a patent on a "Process of Smelting Sulphides," on the basis of experience with this revival of Mr. Hollway's ideas and suggestions, and in June, 1891, was finally allowed letters patent (No. 453,529), covering various claims, by the United States Patent Office.

The patent includes a peculiar, from the practical standpoint, fanciful, construction of furnace, as best adapted to the process, but is not restricted to the use of this furnace. It embraces the use of a heated blast for the purpose of smelting as specified. Locally, much has been made by the uninitiated, in the regions where the process is in use, of this application of a preheated blast as a startling novelty in the smelting of the more precious metals. In truth, however, hot blast, in the course of its nearly sixty years of uninterrupted employment for metallurgical purposes in Europe, has long ago found its way into the smelting of the more valuable metals, and, in fact, is theoretically urged for introduction wherever the nature of the metals treated permits of its use. Thus, it has hitherto been excluded from the smelting of lead on account of the volatility of this metal, or rather, the failure of first trials. The most illustrious example of the long-established use of hot-blast in *copper* smelting in that of Mansfeld. From the point of view of thermo-chemistry, hot blast has advantages beyond the mere heating power of the fuel used to create its heat, and, metallurgically, its superiority has been briefly indicated by Mr. Hollway's statement concerning the greater consumption of siliceous material. In pyritic smelting proper, finally, it has thus far proved an absolute economic necessity, for various reasons due to the behavior of the ores.

Mr. Austin's letters patent contain the following: "The object of my invention is to provide an improved process of smelting applicable to pyrites, sulphides (natural or artificial), sulphurets, arsenical and antimonial sulphurets, zinc-blende, and other ores or artificial compounds of a similar nature. . . .

"My invention relates to the smelting of iron and copper pyrites, zinc-blende, natural or artificial sulphides, and sulphurets of useful metals. Its special object and purpose has been to provide a process of smelting such ores or compounds which, when started, can be maintained or kept up continuously to fuse the cres or compounds, without the admixture of any carbonaceous fuel with the smelting-charge or the application of such fuel during the process within the furnace. As will be seen, I attain the desired end of avoiding the use of carbonaceous fuel during the smelting operation, without employing a bath of molten material, as applied and used in the Hollway and other kindred processes.

"In carrying out my process of smelting, all molten matter is drawn off out of the furnace as soon as formed, and is nct used to generate heat for operating the process.

"The charge described (44 per cent. of sulphide of iron, as contained in iron, copper, arsenical, magnetic, or other pyrites, carrying various percentages of the precious metals, and 56 per cent. of dry argentiferous ores varying in the amounts of silver carried by them) is, by my process, smelted without the use of coke or coal in the furnace, after the preliminary heating, provided the blast of air is heated up to from 800 degrees to 1,000 degrees F., while with blast at the ordinary temperature of the atmosphere, the charge cannot be smelted, but is only roasted.

" . . . No particular form or shape of furnace is necessary to the successful carrying out of my invention. The furnace can be square, round, or rectangular, etc., etc."

The peculiarity of the furnace incorporated in the patent is the presence of a short cylinder, water-cooled or otherwise, made impermeable to heat, which hangs down the center of the furnace-shaft and through which the pyrites is supposed to be fed (by an automatic hopper arrangement gauging the quantity by bulk) independent of the quartz-ores. These are charged in the ring-shaped space, between cylinder and furnace-walls. Contact is thus established at a point near the tuyere-level. The gases and fumes ascend and escape by the ring-shaped space. There is again an auxiliary set of tuyeres, but higher up than in the old Toston furnace. They are here intended for the volatile sulphur alone. This furnace has never yet been built. It is apparently the outcome of speculations on experience at Toston. Needless to say, the necessity of such a mode of feeding for the purpose of bringing the fresh pyrites in contact with the silica at so low a zone in the furnace, in order to avoid the stickiness of the charge, has not elsewhere been recognized, nor endorsed by the experience of others.

Claim number one is the most general and covers the process, independent of the furnace. It is worded as follows: "I claim as an improvement in the art of smelting pyrites, blende, sulphurets, or sulphides, the method of securing continued combustion and smelting without the necessity of the continued use of carbonaceous fuel, which consists in first heating the lower part of a charge containing one, or more, of the substances or compounds up to a point where combustion can take place, and then subjecting the charge to the action of a hot-air blast and continuously drawing the molten products, as formed, away from the path of the blast, substantially as and for the purpose specified."

The rest of the claims all pertain to the mechanical principle of the furnace, *i.e.*, the separate feeding of the pyritic, and the quartz-ores, and may be passed over.

The course of the various experimental and other runs at Toston from 1888 to 1891, appears to have been anything but smooth. No definite and accurate, reliable data concerning either the metallurgical or the economic value of the revived process appear to have been gathered. This was partly due to the fact that the State of Montana is not yet sufficiently developed to be able to insure continuous work on suitable pyritic ores for even one small furnace, and partly to defective apparatus and lack of capital for better applicances. But it also seems to have been owing to lack of technical skill and neglect of chemistry. The patentee left Toston in the summer of 1891.

The Montana and Idaho Mining and Refining Company, to whom the right of patent for the two States included in this title descended, subsequently made a several months' run at Toston under great difficulties as to ore supply. Partly to place the metallurgical side of this enterprise on a more thorough basis, as well as for the sake of a dependent company about to be formed for the use of the process in another part of Montana, the present writer was delegated in August, 1891, to institute a close examination of the merits of the process. Others had slightly looked into it before, with unreliable results.

A short test-run was accordingly made at Toston, with as thorough attention to all details, both financial and technical, as could be brought to bear, and the first accurate facts as to the value of the process from both points of view were thus established. The results were so highly satisfactory and encouraging that soon after the Boulder Smelting Company began operations. Meanwhile the Montana and Idaho Mining and Refining Company followed precedent and kept on running without much difficulty, except as to ore supply, into the winter of 1891–2, until its obligations were fulfilled.

The Boulder Smelting Company built a neat and compact one-furnace plant at Boulder Valley, Montana, which was completed in May, 1892, and blown in in July. Unfortunately, the main ore supply which had been depended upon was far too siliceous to be used in quantity, as the ore ran from 65 to 85 per cent. silica, although the values, chiefly gold, were satisfactory. Purchase ores were very difficult to obtain, so that the plant has lain idle most of the time.

However, the Boulder enterprise set the example for others, with whom the patentee had in the meantime been negotiating. The old La Plata lead smelter at Leadville, Colorado, was fashioned into a crude, temporary experimental plant for the Bi-Metallic Smelting Company, and the feasibility of the process demonstrated by a trial run. Subsequently, the plant was considerably improved, and, being supplied with very favorable ores, continued to run without interruption from August, 1892, to June, 1893. About the date of writing (September, 1893), work has been resumed in a completely remodeled establishment, with three large pyritic furnaces, the sum of $200,000 having been spent on the alteration and enlargement of the works.

Simultaneously with this company, which owns some of the most important mines in Leadville, Colorado, the Summit Mining and Smelting Company put up a similar one-furnace plant at Kokomo, Colorado, not far from Leadville, on the strength of mines of fairly favorable ore at that place. Owing to the singular total lack of copper in these mines, and other causes, some difficulty was experienced there in the beginning. This was corrected by the writer in October, 1892, and the furnace then ran uninterruptedly until June, 1893, when the drop in the value of silver put an end to operations. Work has not yet been resumed.

There are thus, at the present writing, six furnaces in the United States fitted for the process, with a united capacity of nearly 500 tons of ore a day.

TABLE OF PYRITIC FURNACES IN THE UNITED STATES.

Where situated	Kokomo.	Leadville.	Toston.	Boulder.
Number of furnaces	1	3	1	1
Size of furnaces at tuyeres	33″×140″	One 30″×140″ Two 36″×163″	36″×78″	36″×96″
Capacity in tons per 24 hours as run	100	250	40	60

LITERATURE OF THE PROCESS.

The only essay written on the subject of "Bessemerizing Ores in the Blast-furnace," as far as I know, is the elaborate one of John Hollway, of 1879, extensively quoted above.

Copious discussion of the rapid oxidation of sulphur and iron may be found in the earlier days of the Manhès process, but scarce any reference is made to *ores*, *natural* sulphides, as against mattes, or artificial sulphides.

Previous to Hollway, Bode in his treatise on "Sulphuric Acid," in 1872, gave the following temperatures of combustion: Iron pyrites 2,586, pyrrhotite 2,698, protosulphide of iron 2,725, copper pyrites 2,425, copper-glance 1,976, zinc-blende 2,850, bornite 2,246, stibnite 2,517, Mansfeld "rohstein" 2,391, "spurstein" 2,161, and silberrohstein 2,810 degrees Centigrade.

Hollway's figures are: Protosulphide of iron 2,225, sulphide of zinc 1,992, and sulphide of lead 1,863 degrees Centigrade.

Balling, in his *Metallurgische Chemie*, 1882, referring to Hollway's process, finds the temperature for iron pyrites to be 1,719 degrees (1,702 by error).

An article by W. L. Austin in 1887, *Transactions American Institute Mining Engineers*, Vol. XVI., on "Matting Dry Auriferous Silver Ores," only refers to ordinary "Pyritic Smelting," and not to "pyritic smelting proper" as patented by him. Aside from communications to technical periodicals and the inevitable "prospectus," Mr. Austin has not, as far as I am aware, written on this subject.

APPARATUS AND PRACTICAL SCOPE OF THE PROCESS.

The apparatus made use of is scarcely original to the process, except in minor details. Blast-furnace, hot-blast stove, and blower are all that is required, the balance of paraphernalia being that of matte-smelting plants in general. The hot-blast stove is only a semi-novel feature, but the discarding of all roasting apparatus is really a decided distinction from current methods.

The blast-furnace may be of any good type ever employed for matte-smelting, with detail variations fitting it for the application of a heated blast. Its height cannot well exceed ten feet from tuyeres to charging-sill, on account of the density with which the friable, decrepitating ores will pack, if too high. A column of from 8 to 10 feet is also fully sufficient to avoid over-fire. It is best to adhere to the modern use of our external and independent forehearth, and for this reason the slag and matte are to be withdrawn together in a single, continuous stream, and no separation attempted in the furnace itself. The blast may be trapped or not, *i.e.*, the furnace may be built after the type of the "Sumpf," or of the Spur-Ofen, the first being preferable. It is of some advantage to increase the number of the tuyeres; that is, to place them nearer together than is now the customary practice in lead, or copper-smelting. The furnaces and gases may be withdrawn either above or below the charge-floor, the former being better for certain practical reasons due to the physical and mechanical nature of the ores. Except that the bustle-pipe and the branches leading from it to the tuyeres must be adapted to the transmission of highly heated air under pressure, there is nothing required in the constructive details of the furnace at all differing from what individual experience and conviction may dictate as the best form of the modern water-jacket blast-furnace for the smelting of roasted copper ores into matte.

A flue-dust chamber is necessary, as in all blast-furnace work. *A high stack* connected with the same is an essential feature, as the gases leaving the furnace are very obnoxious and injurious to vegetation.

The forehearth may be identical with any of the many varieties in use at copper-matte smelters. Due regard is to be paid, in its construction in detail, to the specific heat of the product made (*i.e.*, its chilling quality) and the rapidity with which it accumulates.

The blowing machinery may be of any kind capable of furnishing the requisite volume and pressure of air. There is no technical objection to the use of blowing engines, which give high pressures, but so far, the ordinary blast-furnace practice has been followed by using the well-known *rotary blowers* of Baker or Root. As a fact, not enough has as yet been done toward increasing the speed of furnace beyond the limits set by the ordinary rotary blower. As the degree of concentration, as well as the amount of ore put through, both depend upon the capacity of the blowing machinery to furnish volume and pressure of air, the confines of lead and copper smelting might well be overstepped to advantage in this respect.

The hot-blast stove is not original to the process, and its use is simply borrowed from iron practice, of course. Nor is its application to the smelting of copper and the precious metals at all a novel idea, as the examples of several European establishments, which have employed a hot blast for decades show. The stove has hitherto proved a necessary adjunct to the chemical basis of the process.

Since the gases from the process are incombustible, and, furthermore, good furnace management will not allow them to escape with any more heat than the gases of a lead furnace exhibit where the latter is in good trim, they cannot be used for the purpose of preheating the blast, similar to the practice in the smelting of iron. The idea of regenerative stoves utilizing the furnace gases is therefore wholly out of the question, and the old-fashioned *cast-iron pipe-stoves* are the only resource, the fuel to be used in these being carbonaceous and the cheapest that is serviceable, of course—wood, petroleum residue, coal, etc. The extreme cost of installation argues against the preferring of generators, producers, etc., for the purpose of gasifying inferior fuel, instead of the direct combustion of carbonaceous fuel in the pipe-stoves. Attempts to consume the *volatile atom of sulphur* contained in the furnace-fumes in the elemental state would be utterly unpractical for various reasons. Very large volumes of fumes and gases would have to be handled, whatever the contrivance might be; not to mention the destructive effect of the gaseous products of combustion on brick, as well as cast-iron. The idea of *utilizing the heat of the slag* wasted at the furnace for the pre-heating of the blast, is much more practical and feasible.

Concerning the actual design of the hot-blast stove it will suffice to say that the process makes no special demands, and that any one of the very many types of cast-iron pipe-stoves which distinguished the pre-regenerative stove period of iron-smelting, will answer, if economical of fuel and effective, *i.e.*, suited to the particular requirements.

The management of the process, as already said, is based on chemical principles, and the skill to apply them metallurgically.

In the matter of the *slags*, which is only the metallurgical way of saying, in the matter of the composition of the charges introduced into the furnace, everything that can be done in lead, copper, or matte smelting in general, can also be accomplished in pyritic smelting proper. As compared with lead smelting, there is a boundless variety of possibilities, without detriment to the character of the work done. Pyritic smelting of either kind is the most adaptable and the widest reaching of all smelting operations in this respect. Virtually, every slag that will run at all at the temperature of the process, can be made, though, of course, this sweeping comprehensiveness must be guided by sound metallurgical judgment. The amount of silica in the slags may vary from below 28 per cent. to above 48 per cent., to suit the ores treated, the required degree of concentration, etc. If the ores are sufficiently rich in iron, the use of lime-flux is not obligatory, but when there is a deficiency of iron, the lime in the slag may be run up to 30 per cent. A deficiency of iron in the ores, however, may always be corrected by using matte previously made.

Thus, in respect to the variety of slags which may be made in order to suit existing combinations of ores, the process is as tractable as any branch of matte-smelting Still, it is always *limited to the presence of sufficient sulphur and iron*, on the one hand, to maintain fusion without the auxiliary addition of too much coke or other carbonaceous fuel, and, on the other, without raising the concentration of the ore into matte to such an extraordinary degree that the minute amounts of matte will remain entangled in the slag and not separate, or that the sulphur is actually all oxidized and no matte made at all.

It is readily possible to make the slags uniformly and continuously satisfactory in point of values retained or entangled. The most severe method of testing them, by taking as an average sample that obtained by catching a small ladle or spoonful every quarter of an hour off the full stream of slag which leaves the forehearth, and treating the whole as the sample of that shift or period, is employed. The records compare most favorably with the assays in the case of lead and copper furnaces, and when very rich ores are smelted, are superior. The slags seldom go above 1.5 ounces (0.005 per cent.) of silver per ton, no matter how rich the matte made is, as proper metallurgical precautions may be taken to keep the values in them down very low. Copper is a great necessity for this purpose, particularly when there is considerable zinc present.*

Except occasionally, and for specific purposes, a slight percentage of coke, *i. e.*, from 1½ per cent. to 2½ per cent. of the weight of the charge has hitherto always been used, even with the best ores. The action of this small amount of carbonaceous fuel seems to be more *mechanical* than of thermal advantage, however. That is to say it increases the porosity of the charge. Where local conditions counsel it, it is advisable and possible to use cordwood for this purpose instead of coke. It need simply be sawed into blocks, and will thus prove of better service than when used in the form of charcoal. There is no particular reason why a more energetic activity of furnace than has hitherto been possible at any of the works, which a more powerful blast would ensure, should not in the case of coarse and massive ores at least, allow of the continued exclusion of all carbon. A deficiency of sulphur may be corrected by a greater use of coke, as may also too low a temperature of the blast. Still, to correct the latter error by this means is very faulty, since there is a wide economic difference between the chemical action of the same amount of heat out of cheap fuel, as transmitted to the blast in the stove, and out of coke in the furnace, inasmuch as the presence of carbon in the furnace lowers the degree of the concentration of ore into matte.

* It will be noticed that Mr. Sticht is considering the subject as a silver and gold smelter, rather than a copper smelter pure and simple.

With a given plant, the *rate of concentration* can be regulated at will; that is, it is possible to set out to make the amount of matte made out of the ore bear any desired ratio to the amount of the ore itself. This ratio may vary from 2 to 1, *i.e.*, 2 parts of ore turned into 1 part of matte, up to 20 to 1. The degree is purely a matter of economical judgment and metallurgical manipulation. Matte which is not high enough in value to be worth shipping may always be made richer by re-concentration; that is, by sending it through the furnace again, either alone or with ore. As a rule, however, it is preferable to strive for the necessary richness of the matte at once, and not to put a finished product into the furnace twice, or oftener, because the percentage of recovery in consecutive smeltings of the same substance decreases in a geometric ratio.

The facility with which the process is conducted depends to an important extent on the physical or mechanical character of the ores. Iron pyrites has a marked tendency to decrepitate with heat, in consequence of which fact the column of ore in the furnace is apt to pack tight and dense, and cause a stoppage of all fusion by obstructing, and finally preventing, the passage of the blast through the material. For this reason, very high furnaces are entirely out of the question, even in the case of very coarse ore. It is simply a matter of the time required for coarse pieces to decrepitate to a point of impeding fineness during their descent in the furnace, before fusion would begin. Other sulphide ores decrepitate less than iron pyrites; but the avoidance of trouble from this quarter is an important point in the treatment of most of them. If the column of ore exceeds a certain height, the condensation of the volatile atom of sulphur in the upper regions of the furnace will also give much trouble. The same again is true when disturbances occur in the proper relative distribution of the various zones of the furnace above one another. When the fire creeps up, the ready fusibility of protosulphide of iron, and its equal readiness again to chill, may cause the top of the charge to stick together to such an extent that the descent of the material is wholly prevented. Every means must therefore be used to keep the charge as porous and loose as possible, to prevent the furnace from choking. As this question is simply one of the merest mechanical difficulties with an ore, it has absolutely no bearing on the true metallurgical value and feasibility of the process. Experience in practical furnace manipulation will enable one to meet these difficulties, which, to a certain extent, characterize all matte-smelting in the blast-furnace, or better, to prevent them from asserting themselves. It is best to use the ores as coarse as possible and not to crush or handle them any more, previous to smelting, than is really necessary. It is, of course, possible to use a quantity of very finely divided ore, say pyritic concentrates, but the bulk of a charge can scarcely consist of such; more than 50 per cent. of fines is hardly advisable if satisfactory metallurgical recoveries are desired; and the exclusive use of fine concentrates is impossible. When running on such, the column in the furnace perhaps could not be kept higher than say 2½ feet to 3 feet, and this would entail extravagant losses by volatilization and flue-dust, and by scorification. Solid, massive ores, which break in good-sized chunks of apple-size and above, with only the fines unavoidably made in mining and crushing, are the ideal material. Concentrates may be bricked, but the local conditions are seldom such as to warrant this, on account of the extra cost, as well of the bricking operation and apparatus as of the smelting of the barren binding material.

The pressure of the blast at any of the works now in use has probably not exceeded 1¼ to 1½ pounds to the square inch, measured at the furnace and not in the blower-room. It has hitherto been quite essential to pre-heat the blast; temperatures of not more than 305 degrees F. even showing a decided improvement over a cold blast in the running of the furnace. Besides exercising a more energetic oxidizing action and thus facilitating the process in general, the tendency of a heated blast is to concentrate the heat more in front of the tuyeres and to avoid the agglutination of the charge higher up. At the same time, as the furnace column may then be lower, the volatile atom of sulphur is more effectually sublimated and prevented from condensing, while that part of it which seems to get down even to the vicinity of the tuyeres, is also burned. The temperatures of the blast used vary from 600 degrees to 1,200 degrees F. (320 degrees to 605 degrees C.). As far as the working of the furnace itself is concerned, there is no limit. Considerations of expense usually control the choice of the temperature adhered to. According to the locality, etc., this item varies from 10½ cents to 25 cents per ton of ore smelted.

With proper management, the campaigns of the furnace may be as long as those of any furnace run on the ordinary methods of matte smelting, with or without much copper. Zincky ores may cause transient stoppages, for the purpose of clearing the furnace-throat accretions, but this is done without blowing out. The discreet use of slag-charges will pull the furnace through many a difficulty which would otherwise cause a shut-down. The outside forehearth will occasionally have to be replaced by one freshly prepared, but there is no circumstance in the conduct of the process so insurmountable as to compel the complete suspension of work from time to time. The Colorado furnaces ran uninterruptedly for 9 and 11 months, finally being stopped from reasons independent of any peculiarity of the process.

The amount of ore smelted will, of course, vary with the size of furnace and other conditions, including the character of the ore as to size and decrepitation. The Kokomo furnace, which is 33 inches by 140 inches at the tuyeres, has averaged 104 tons of ore in twenty-four hours for a month at a time, besides a large proportion of fluxing-slag and limestone. This great tonnage, however, is commercially a mistake, inasmuch as the losses by volatilization and in flue-dust were high at the same time. The three furnaces at Leadville, one of which is 30 inches by 140 inches, while two are 36 inches by 163 inches, put through about 250 tons of ore a day. This, again, is low for furnaces of this size. It is accounted for by the lack of sufficient air, and a low temperature of blast.

Duly considering the above data, the statement that the actual cost of smelting may readily be kept below $4 a ton of ore and be made to approach $2 a ton, will surely be credited. These figures stand for Montana and Colorado, and include all expense of the enterprise.

The metallurgical losses have hitherto varied from 5 to 15 per cent. loss on the silver to a loss on, or plus of, 5 per cent. on the gold. It has not up to date been agreeable to the works in Colorado, who alone have been running continuously, to lay stress on accurate determinations of their recoveries, since these involve some expense through extra labor and loss of time, and the knowledge is in part only curious. Hence, their figures are open to doubt, as being too bad. The attempt to work up great quantities has led the works to be satisfied with poor metallurgical work, and, on the other hand, the balances have been hastily and incompletely drawn up, *i. e.*, credit material has been neglected. It does not seem at all reasonable that long continued runs should show up losses of as much as 18 per cent. of the silver, whereas short trial campaigns gave only 5 to 7 per cent. loss. The advantage of better recoveries lies entirely on the side of the long campaigns. For reasons of first cost, flue-dust chambers, etc., were left out all of the works; and this fact alone is, to an important extent, the reason of the losses as given in the higher figures. Still, in the very face of all this laxity on so serious a question, the smelting losses cannot be said to be discouraging. On the contrary, those accustomed to practical metallurgical work will agree that the showing as given in the following (see recoveries and cost) is, under the circumstances, entirely satisfactory. The best recoveries hitherto obtained in any single campaign were 95.3 per cent. of the silver and 101.6 per cent. of the gold.

The degree of the recovery is influenced by the presence of *deleterious substances* in the ores treated, similarly to other varieties of blast-furnace smelting. The process has the usual sensitiveness to the effects of zinc-blende, when the amount of the same is appreciable. In this case, it is hopeless to base expectations of utility on the fact that this substance is likewise combustible. There is no doubt that the combustion of zinc-blende, in an amount sufficient to maintain fusion of the charge, would entail losses of the precious metals to an alarming extent. The same is true of sulphide of lead, or galena, ores. The combustion of both is accompanied by the volatilization of silver. The blende could be thus utilized with more powerful facilities for blast, or in very low furnaces. Under existing conditions, it is only partly oxidized, and enters the slag to that extent, and is also deposited in the furnace-throat. A great portion of it, in its unaltered chemical state, is held in solution in the slag chemically, or merely mechanically. Another portion enters the matte unaltered; and the balance, except what escapes in the fumes, is sublimated and regenerated in the upper, cooler regions of the furnace shaft. Its immediate action is to vitiate the separation of matte and slag by decreasing the specific gravity of the former. The slag will therefore run high in values. The only practical remedy is to resort to the use of copper ores, for the purpose of both increasing the specific gravity of the matte and, also, apparently, of extracting the precious metals from their combination with sulphide of zinc, by means of their greater affinity for sulphide of copper. Copper thus becomes practically indispensable to pyritic smelting of both kinds, yet very minute amounts, *i.e.*, 0.75 per cent. on a charge, will suffice to increase the virtue of sulphide of iron as a collector of values.

Lead may also effect a closer saving in the matte. Still, considering that it will occur as a *sulphide of lead* (*galena*), and that this is disposed to distil or sublimate, with attendant losses in precious metals similarly to blende, though at a lower temperature, its advantage is very doubtful. It is hopeless to expect to save a satisfactory amount of the lead when present as a sulphide, except in the fumes. There is good reason to believe that *carbonate of lead ores* might be utilized to better advantage than galena for the purpose of collecting the precious metals in *bullion*, as the reducing action of the furnace may be sufficient to accomplish this, relying on the reducing nature of the sulphurous acid in the atmosphere of the furnace. In the case of galena, the lead-loss appears to be tremendous, having been determined to be 60 to 70 per cent. in the only instance where the subject was investigated; but it must be remarked that the amounts of lead then dealt with were extremely small. There is also reason for believing that when there is sufficient sulphide of lead present, there is a reaction between part of it which is oxidized and part which is not. Experience at least, proves that there is a formation of metallic lead, besides scorification of quantities as an oxide; distillation of probably larger amounts as a sulphide, and loss of the same in the fumes; and condensation of the balance as regenerated sulphide in the furnace-throat. Some sulphide also enters the matte. The collecting power of this metallic lead is extreme, especially for gold. The lead bullion obtained in small amounts in both kinds of pyritic smelting is always exceedingly rich, but, as alloys of this kind are very unwelcome for practical reasons, lead offers no inducements, at the best, as a collector of values in comparison with copper. As an antidote to zinc-blende, or for cleansing the slag, it has no virtues whatever. In itself, the scorification of lead, if effected, does not necessarily drive silver into the slag; but lead in small amounts, in the presence of other sulpides which it is expected to extract the silver from, is not a factor to be relied upon, and the disposition of sulphide of lead, of whatever origin, to carry off silver by volatilization is a further serious fault.

As already said, the strong mutual affinity of the sulphides of silver and copper permits of the use of a very small proportion of the latter. As a natural result, in such a case, while the silver recovery is good, the copper loss will be high and may amount to 25 to 30 per cent. Pyritic smelters, treating silver and gold ores chiefly, with but very little copper, are obliged to ignore this loss, regarding the few tenths of one per cent. of copper on the charge as a mere fortunate accident; and purposely do not increase the percentage. Thus they strive for a very high degree of concentration, say 25 into 1, and the copper even then will save the silver and gold though half of itself may be lost.

There is no limit to the variety of the *chemical composition* of the ores which may be treated with the fundamental pyritic ore. Ores containing any amount of silica, alumina, zinc oxide, lime, magnesia, baryta, iron, manganese, etc., or any combination of oxides, may be used, and are exactly as suitable as for ordinary lead or matte smelting. In two instances, however, the process is superior to these, and these are the removal in the blast-furnace of the arsenic in the crude ores, and the positive advantage to be derived from so obnoxious (usually considered) a substance as sulphate of baryta (heavy spar). The former, of course, is present only as a sulphide (or in connection with sulphides which readily part with some of their sulphur), and always distills as such as soon as a newly fed charge is warm enough after striking the top of the ore-column in the furnace. The action is quite instantaneous. It is thus completely removed before it can reach a zone where its affinity for copper and iron would again assert itself. *Excessive* amounts of arsenic, it is needless to say, will lead to the formation of speiss in a rapidly working furnace; but the presence of ordinary quantities of mispickel or other arseniureted ores has no such effect.

As regards *heavy spar*, it must be remembered that the laboratory of the furnace is an oxidizing one; the reducing power of the sulphurous acid gas, at all events, seems insufficient for the purpose, so that there is *not* a reduction of the sulphate of baryta to sulphide of barium, which substance would work havoc in the separation of matte and slag; but, on the contrary, a formation of easily fusible silicate of baryta through the decomposition of the spar by silica and the liberation of the sulphuric acid and the baryta in the same, from each other. This reaction takes place in *all* smelting operations, whether in blast-furnaces or reverberatoires, where silica is present and the working atmosphere is not of a nature to reduce the sulphate.

The *gases* coming from the furnace consist mainly of nitrogen and sulphurous acid, with some sulphuric anhydride, the first named greatly predominating. Also small quantities of carbonic acid, and oxide; hydrogen (?) and (possibly) free oxygen. The reactions are complete; unstable oxidizable compounds like the sulphides of carbon (and hydrogen) do not assert themselves. The volatile atom of sulphur liberated from the iron pyrites, and whatever sulphides are sublimated, like those of arsenic and lead, of course accompany the above as fumes; also the moisture of the ores. The gases have never been analyzed. From results obtained by Professor Frankland in 1878, during Mr. Hollway's famous experiments, it would follow that pyritic smelting proper makes less gases than either lead or copper-smelting in blast-furnaces (based on relative percentages of nitrogen) for the same amounts of air blown in.

In the cases where a good deal of iron pyrites is smelted, every new charge fed into the furnace gives off dense yellow fumes of metallic sulphur, more or less darkened to orange by the sulphides of arsenic which are cut short abruptly when all the loose sulphur is volatilized, and are then succeeded by a brilliant white stream of sulphurous gases.

This play of colors is a most striking and unwonted phenomenon, and gives the process a very unique feature. When ores, or material containing no loose sulphur are smelted, the current of gases is uniformly white.

The amount of flue-dust made seems to be smaller than in ordinary operations in the blast-furnaces; a fact directly related to the smaller quantity of gaseous products of combustion furnished by the process, as remarked above. The top of the charge being less agitated by the passage of gases, there is less mechanical dragging along of fine particles.

THE RECOVERIES OF VALUES.

It is only justice to the process to say, at the outset, that I have some professional hesitancy about giving publicity to the following figures, because they have to be submitted accompanied by excuses and explanations as to why some of them are not better.

The best results yet obtained anywhere were those of my own test-run at Toston, Montana, two years ago. Considering that there was a loss by flue-dust (which was not caught, owing to the absence of the usual appliances for that purpose), the recoveries were actually very good and fully satisfactory. We there also ran at a comfortable pace only, and did not follow the principle of putting through a great quantity of ore, regardless of losses, and for this reason, no doubt, the results were so remarkable.

Aside from this determination at Toston, the Boulder, Montana, works were the only ones which always pursued the accurate establishment of their metallurgical losses with every care and attention necessary for the purpose. Unfortunately, ores not suitable to the process, and defective machinery, vitiated the results, as further explained below.

I have been vainly hoping that the two works in Colorado would see fit to draw up comprehensive metallurgical balances, but, as I am informed by private advices from the metallurgist in charge at Kokomo, no *exact* determinations of losses were ever made at that place. And the very unsatisfactory data which I have received from the management at Leadville plainly show that here, likewise, this subject has been neglected. Nor will those who are acquainted with local conditions at these two places be at all surprised that this is so. After the first actual demonstration that the process would "work," and the determination of the losses and the cost per ton under the conditions of this first demonstration, the interest in the question ceased. This first investigation in Colorado was conducted at Leadville and the results are given below. Subsequently, the aspiration was simply for *output*, and the question of *saving* dwindled mainly into the diminution of the labor item. With the exception of the gentleman who personally ran the Kokomo furnace for a month or so, after my own investigation and correction of the troubles there a year ago, there has not been an educated metallurgist in charge at Kokomo, and the Leadville works have never been under the supervision of such an one. All this is to be deplored, because these Colorado establishments have put through a great quantity of ore in long, continuous runs, and a long campaign is best suited to the purpose of determining the recoveries.

The figures received from Kokomo, Colorado, vary in the recovery of silver from 90 per cent. to 85 per cent., and in the gold from 100 per cent. to 103 per cent. Copper and lead were not taken account of. The principal pyritic ore at the place carries about 42 per cent. iron and 4 per cent. silica, is very crumbly and frequently degenerates in the mine into a coarse, granular mixture of iron pyrites, zinc-blende, and galena, the last two occasionally going up to 20 per cent. each. There is a most singular total lack of copper in the large pyritic veins of the district, hence ores carrying this metal must be purchased from afar and are, therefore, used in very small proportions only. The recent introduction of flue-dust chambers (of size and construction unknown to writer) is reported to have increased the silver-saving from 85 per cent. up to 90 per cent. This recovery includes matte-concentration (in the same furnace, together with ores) which was periodically resorted to, for the purpose of decreasing freight and treatment charges, (by shipping the same values in a smaller tonnage of matte), and which heavily increased all metal losses. The forced capacity of over 100 tons of ore a day (in one furnace) led to the bad practice of using a very high blast on a low column of ore.

I have never received any figures from Leadville, Colorado, subsequent to the results of the preliminary test-run made under the direction of the patentee, which I would dare or care to give as the result of work doing honor to the process or justice to its character. But even this trial-run was made under such singular metallurgical supervision that I find myself constrained to ask for proper allowances to be made in criticising the results. It will suffice to say that the ores were not analyzed, but their composition, so far as it was taken into account at all, was only guessed at. The run was, therefore, not an exemplary one. It was fully controlled by the contracting parties, however, as far as the financial side, etc., was concerned. Ignoring the patentee's own figures, those of the leasers are as follows:

Average rate of concentration	9.85	ore into 1 matte.
Tonnage (of the 36-inch by 80-inch furnace)	60	tons of ore a day.
Cost per ton of ore (high on account of unnecessary labor)	$4.49	
Silver recovery	93	per cent.
Gold recovery	95	"

There was no flue-dust recovered (no chambers, etc.).

The slags were very basic (25 per cent. silica) and high in values. The pressure of blast was 4½ inches, equal to 36 ounces as registered in the blower-room, some distance from the furnace, connection being made by a rather leaky blast-pipe; and the column of ore in furnace was only 5 feet. Making due allowances for work as bad as this, the leasers saw sufficient encouragement in the above figures to accept the financial lucrativeness of the process as established.

The mines belonging to this company, and utilized for the process, are among the most famous and largest in Leadville. This trial was made in the spring of 1892, and several hundred tons, at least, were smelted (close to one thousand, I believe).

The trial run at Toston, Montana, under my own supervision, was not very long, scarce 300 tons of ore being treated, but sufficiently extensive to demonstrate to us the various advantages and the metallurgical feasibility and precision of the process.

The condensed statement of recoveries is as follows:

SMELTED.

Dry weights—							
Ore	557,847 lbs.,	containing	2934.08 ozs.	silver and	175.10 ozs.	gold.
*Slag	155,775 "	"	124.62 "	"	...... "	"
Totals			3,058.70 "	"	175.10 "	"

PRODUCED.

Matte	84,590 lbs.,	containing	2,346.29 ozs.	silver and	113.52 ozs.	gold.
Furnace returns.	74,855 "	"	561.80 "	"	53.71 "	"
Lead bullion,...	100 "	"	7.87 "	"	10.59 "	"
Totals			2,915.96 "	"	177.82 "	"

* From an old lead dump.

N. B. There was no recovery of flue-dust (no appliances).

The rate of recovery on silver was 95.3 per cent.; on gold, 101.6 per cent.; concentration, 6.6 ore into 1 matte.

The slags averaged about 1 ounce in silver (0.0034 per cent.) and a trace in gold.

The furnace treated only about 35 tons of ore a day.

The slags made had to be very siliceous on account of a scarcity of iron pyrites. Hence, there was also an abnormal percentage of coke used (from 3 per cent. to 6 per cent., figured on the ore). The slags contained from 41 per cent. to 48 per cent. silica.

All conveniences at the works for transportation, etc., were extremely crude, and much hand-labor was required.

There was also an appreciable loss in consequence, through the scattering of ore and matte, also through poor appliances on and about the furnace.

Types of the ores smelted are:

Percentage of iron	19.2	30.0	20.3	35.2	33.0
Percentage of silica	40.0	29.0	40.6	15.3	14.0

Among the 557,847 pounds of ore smelted, there was quite an amount of bricked concentrates, and also a quantity of barrings (furnace returns). It is but fair to count the latter, as well as the clayey binding material of the bricked concentrates, in with the ore smelted, as I have done; for a determination of recoveries and cost, for the purposes of a test, cannot properly concern itself with anything but the chemical suitability of the material treated, independent of its derivation.

The 155,775 pounds of slag smelted were fluxing-slag and taken off the dump of the plant (formerly a lead-smelter), and were duly sampled.

Zinc-blende and galena were present to some extent, as also was copper pyrites. The galena causes the formation of a rich gold-silver-lead alloy, which is taken account of above as "bullion." It assayed 157.4 ounces of silver and 211.80 ounces of gold to the ton. The assay of a small amount of rich lead dripping through the drop-bottom of the furnace was 178.2 ounces silver and 276.60 ounces gold to the ton.

Limestone was used on the charge to help out the deficiency in iron, *i.e.*, to allow the supply of pyrites to last longer and to smelt more siliceous ore.

No account was taken of the small amount of copper present, although it materially assisted the recoveries.

The amount of furnace returns seems heavy in comparison with the matte made. They consisted of the barrings left in the furnace after blowing-out. The quantity would have been about the same, hardly greater, had the test-run lasted much longer than it did (eight days).

The Boulder, Montana, establishment met with difficulties, not inherent in the process, from the start.

A leaking stove, (for the hot blast), which was but poorly repaired from campaign to campaign, caused frequent complete shut-downs. Every time this occurred silver was unavoidably lost by volatilization in the blowing-out operation, as always, and the small amount of ore put through is thus often charged with losses of an ordinarily infrequent kind. The great lack of suitable ores caused an extreme use of limestone (to supplement iron) and this greatly reduced the amount of ore put through, and, moreover, demanded indirectly an increased consumption of coke. Zinc had to be contended with in abundance, as also galena, both augmenting the loss of silver by volatilization. No flue-dust was saved, the plant lacking the requisite appliances. Matte had frequently to be used over again for the sake of its iron and sulphur. Altogether, the series of campaigns was a continued struggle with the very worst and most unsuitable conditions under which the process could be attempted, and the recoveries bear witness to this fact in the silver losses. However, the loss was not in the slags, for these were uniformly satisfactory (except during casualties, as usual), seldom exceeding 1.5 ounces (0.005 per cent.) in silver, and generally carrying only 0.4 to 0.9 ounces in this metal, the gold being a trace at most.

The loss was in reality from the top of the furnace, not from the bottom.

A survey of the ores smelted, by averages of mixtures of from 60 to 350 tons, will characterize the situation:

Silica. Per Cent.	Iron. Per Cent.	Zinc. Per Cent.	Copper. Per Cent.	Lead. Per Cent.	Company's Mine.
60.26	15.64	1.18	0.65	....	Boulder.
56.81	17.38	3.91	1.30	....	Boulder.
24.12	23.84	3.43	8.28	....	
23.81	27.77	5.80	4.03	....	Boulder.
22.61	31.73	....	....	....	
19.74	27.42	6.31	1.82	....	
15.35	21.17	4.30	1.02	....	
8.28	18.16	20.94	3.49	10.8	Butte ore.

The first two were the staple ores from the company's own mine, and the last was the most abundantly obtainable, and had to be worried through with the rest, giving much trouble.

Add to this, the effect of frequent blowing out on account of the failure of the stove to transmit the blast, and the silver-loss becomes plain.

Several attempts were made to run with a cold blast, for which the stove could be disconnected. The ensuing disastrous results are also embodied in the following statement:

Pyritic campaigns ....2,412 charges.

SMELTED.

	Dry Weights Pounds.	Gold. Ounces.	Silver. Ounces.	Copper. Pounds.	Lead. Pounds.
Ores................	1,524,061	351.324	9,398.38	26,747	42,740
Matte	268,440	202.534	5,088.42	20,105	6,228
Totals..........	1,792,501	553.858	14,486.80	46,852	48,968

PRODUCTS.

Matte..............	274,392	516.038	11,307.02	38,839	7,851
Furnace returns.....	250,714	68.980	1,179.29	6,883	10,845
Totals...........	525,106	585.018	12,486.31	45,722	18,696

N.B. There was no recovery of flue-dust (lack of chambers, etc.).

	Gold. Per Cent.	Silver. Per Cent.	Copper. Per Cent.	Lead. Per Cent.
Rates of recovery	105.63	86.19	97.59	38.18
Rate of concentration........	13.5 crude ore into 1 matte.			

The very slight loss of copper, with an average, in the ores, of only 1.7 per cent. of this metal, is very gratifying. In the total material treated (ore plus matte, etc.), the copper averages 2.6 per cent., and the lead 2.7 per cent. The extraordinary lead-loss is instructive, for there is no doubt it was directly influential in increasing the silver-loss. Samples of flue-dust taken off the roof near the furnace chimney proved the dust to be the richest in lead of all materials made or handled.

The following statement includes the campaigns of the one above and also the results of a series of campaigns made with an admixture of oxidized ores, and with the use of the ordinary percentage of coke, *i.e.*, runs made on the plan of common pyritic or matte-smelting. This had finally become a matter of necessity with us, on account of the *lack of pyrites.* The concentration was driven very high at the time for a relatively long period, as high as 23.6 into 1, the accompanying copper-loss amounting to 21.5 per cent., by correct determination, because of the very slight percentages dealt with.

Mixed campaigns: 2,704 charges "pyritic;" 1,538 charges "ordinary."

SMELTED.

	Dry Weight. Pounds.	Gold. Ounces.	Silver. Ounces.	Copper. Pounds.	Lead. Pounds.
Ore..................	2,434,300	565.486	13,295.89	52,174	64,651
Matte...	353,157	238.046	5,615.59	22,093	9,999
Totals..........	2,787,457	803.532	18,911.48	74,267	74,650

PRODUCTS.

Matte................	319,872	682.024	14,543.70	53,914	9,225
Furnace returns.....	461,204	141.548	333.78	13,326	13,794
Totals..........	781,076	823.572	16,877.48	67,240	23,029

Rates of recovery, per cent..... 102.49 89.24 90.54 30.85
Rate of concentration.......... 16 crude ore into 1 matte.
N.B.—Flue-dust not taken account of.

For reasons given above, the copper-loss (9.46 per cent.), was greater than in the "pyritic" campaigns.

There was no opportunity to work up the very low grade furnace returns (mostly barrings) from the numerous short campaigns, hence the great quantity of such returns under "products."

The average assays of the ores smelted and the matte made are as follows:

	Gold, Ounces per Ton.	Silver, Ounces per Ton.	Copper. Per Cent.	Lead. Per Cent.
Ore	0.465	10.92	2.14	2.66
Matte	6.373	124.20	23.12	4.00

The average excess of iron over silica, in the ores which had such an excess, was only 10.9 per cent. and the average of all the ores received at the works during the period covered by above statements shows an excess of 17 per cent. silica over iron.

As said before, I regret exceedingly that the results at Boulder, especially the silver recovery, were so seriously influenced by local difficulties, and trust due consideration will be given the latter in judging the process by the figures obtained. My sole reason for disclosing these is to avoid possible misinterpretations of silence in this respect.

In my own professional estimation, there is nothing in the chemistry of the process to invest it with wasteful features when the ores are reasonably suitable.

There is a remarkable plus in gold in all of the above statements. In Colorado much of this must be accounted for by the custom of calling less than 0.1 (0.00034 per cent.) of an ounce of gold a "trace" in the assay reports and ore settlements. At Boulder, every fraction of an ounce of gold found by assay in the ores, below 0.1, was taken into account in the statements.

But even actual traces of gold, not to be weighed in the assay of the ores, join the matte, the concentration of which makes the gold appreciable.

The silver-loss, I am confident, can be kept from overstepping the 5 per cent. mark, with proper metallurgical care and suitable flue-dust chambers, etc. In the gold, one would always be inclined to expect a slight gain.

THE COST OF SMELTING.

The cost per ton of ore smelted during the Toston trial-run was determined to be $4.60, as follows below. This must be considered rather high and purely local, for reasons given.

Toston test-run from 7 P.M., August 21, to 7 P.M., August 29, 1891, including stoppages and work of blowing-in, also part of blowing-out (278.923 tons of ore smelted).

	Totals.	Per Ton of Ore.
Labor (blast-furnace, sampling, general, etc.)	$732.53	$2.63
Salary (local supervision)	60.00	21
Coke for blast-furnace	190.10	68
Coal for motive power and hot blast	224.00	80
Limestone for blast furnace	35.58	13
Cartage of material about the plant	13.33	5
Coal oil and supplies for motive-power and works	20.00	7
Assay office supplies	4.00	1
General office expense, express, telegrams, etc.	5.57	2
Total	$1,285.11	$4.60

Concerning this cost per ton, it must be said that the furnace was small (36 inches by 78 inches), that there were frequent stoppages, that the speed of running was not forced, and that the presence of a large proportion of fines, coming from the poorly bricked concentrates, retarded the furnace. Hence, only 35 tons of ore per day were put through, whereas the same furnace gang could easily have handled 60 tons. The works are in a dilapidated condition and are very awkwardly arranged for transportation of materials. The labor item, therefore, is much higher than would be the case in a well-designed plant. The item for coal is also about once again as high as need be, on account of an erroneously constructed fire-box on the stove, which compelled the use at times of as much as 10 tons of coal a day ($35), where 5 or 6 tons ought to have been sufficient. More coke was required than ordinarily would be the case, first, on account of its very poor quality (Montana coke with 18 to 25 per cent. ash), and then because of transient difficulties with the new process.

Taking all this into consideration, the cost was very satisfactory to us, and it was evident that a well-arranged and well-constructed plant, which avoided all superfluous hand-labor, would materially reduce it.

The daily cost-sheet at Boulder, Montana, was as follows:

Labor—2 furnace-men at $3.25	$6.50
2 feeders at $3.25	6.50
4 pot-pullers at $2.75	11.00
4 feeders' helpers at $2.75	11.00
2 weighers at $2.75	5.50
4 wheelers at $2.75	11.00
2 firemen at $2.75	5.50
2 engineers; 1 at $3.00, 1 at $2.75	5.75
2 foremen; 1 at $4.00, 1 at $3.50	7.50
6 laborers at $2.25	13.50
1 night watchman at $2.25	2.25
1 blacksmith at $3.50	3.50
1 blacksmith's helper at $2.50	2.50
1 assayer's helper at $2.50	2.50
	$94.50
Salaries—1 assayer and chemist	$5.00
Superintendence and management	36.66
Supplies—For blacksmithing, illumination, assay office, motive-power, etc	5.00
Coke, 3½ tons at $12.00 delivered	42.00
Limestone, 15 tons at $1.30 delivered	19.50
Cordwood for stoves (750° Fahrenheit), 3 cords at $3.50 delivered	10.50
Cordwood for boilers, 2 cords at $3.50 delivered	7.00
Total	$220.16

The amount smelted varied considerably with the ores. These have already been characterized. The average amount when the stoves were in good trim and allowed of a pressure of 12 ounces at the furnace, was about 55 tons of these ores, together with 15 tons of limestone, 3½ tons of coke and 20½ tons of fluxing slag. Thus, while the proportion of coke to ore appears very great, the quantity of dead and barren material made it obligatory. Under these circumstances the cost of smelting was unavoidably high, being $4 per ton. Of this $1.72 fell to labor, $0.76 to coke, $0.35 to limestone and $0.19 to heating the blast.

With good ores, the furnace would readily have smelted 70 tons of ore, with 18 ounces pressure of blast and only 2½ per cent. of coke (with 20 per cent. ash in it). If, at the same time, limestone might have been totally avoided, then the cost per ton, labor, and all other items remaining exactly as given above, would have been only $2.57. Unfortunately, this point could not be reached by us, although occasionally, for a shift here and there, the cost did not exceed $3.50 per ton of ore.

Still, this is not doing as well as may be and has been accomplished for long periods of time elsewhere, as the following example of Kokomo will show. The figures for Leadville must be substantially the same, as the conditions are virtually the same in all respects at both places.

The daily cost-sheet at Kokomo, Colorado, is the following:

Labor—2 furnace-men at $4.00	$8.00
2 tappers at $3.00	6.00
4 pot-pullers at $3.00	12.00
4 feeders at $4.00	16.00
6 wheelers and weighers at $3.00	18.00
3 engineers; 2 at $4.00, 1 at $3.50	11.50
2 firemen for stove at $3.00	6.00
2 foremen; 1 at $5.00, 1 at $4.50	9.50
12 laborers at $2.50	30.00
1 assayer's helper at $3.00	3.00
1 blacksmith at $4.00	4.00
1 blacksmith's helper at $3.00	3.00
	$127.00
Salaries—Clerks, etc.	$10.00
Assayer	5.00
Management, superintendence, etc	unknown.
Supplies—Coke, 4 tons (Colorado), assumed high, at $6.00 delivered	24.00
Limestone, 25 tons at $1.80 delivered	45.00
Petroleum residues for firing (640° Fahrenheit), 10 barrels. delivered at stove at $1.10	11.00
Same for boilers, 20 barrels	22.00
Sundries	unknown.
Total daily expense	$244.00

For 100 tons per 24 hours, which has been the average forced capacity of the Kokomo furnace for weeks at a time, this makes $2.44 per ton of ore, and allowing $0.35 for probable proportion of items not specified above, $2.79 as the current cost of smelting one ton of ore. For an actually more remunerative tonnage, *i.e.*, one involving less metal-loss, say 85 tons of ore a day, the cost, taking limestone and coke in the same ratios as above, would be \$3.16. If the use of limestone in this case could be abandoned, the furnace would again readily put through 100 tons of ore, and without detriment to the recoveries.

The slag at Kokomo is usually carried off with a strong jet of water, so that the duty of "pot-pullers" is dispensed with.

The number of laborers is also usually smaller than given.

COST OF WORKS.

The cost of a one-furnace plant in the Rocky Mountains may be inferred from the following condensed statement of the cost of the plant at Boulder, Montana, which is the best arranged and constructed of the four works.

The building is merely a skeleton frame of lumber with a covering of boards and shingle roofing, but economy was not the sole reason of this.

It is a fact that iron work suffers very rapidly from the fumes escaping from the throat of a furnace where there is no high stack connected with the latter, for the purpose of leading the gases to a high elevation above the ground. Hence, iron is best avoided in the construction of the building, particularly sheet-iron roofing and siding, and especially in localities where the rainfall is heavy.

In the latter case, even a high stack may not be a safeguard against speedy corrosion. The hot-blast stove at Boulder is of a very efficient horizontal type with concentric pipes. Unfortunately it is not well put together, and leaks, and was never sufficiently repaired.

The stoves at all of the other works are of the well-known hanging U-tube type, and are rather cramped in dimensions for the duty expected of them.

The Boulder furnace was designed for the extraction of the gases and fumes below the charge-floor and is built entirely of iron and steel, having only about 1000 fire-brick, (and no red brick) in its construction. It is water-jacketed from hearth to throat, and though designed independently, happens to resemble the lead furnaces of the Zeehan and Dundas Works, Tasmania, the upper jackets being similarly suspended from a frame supported by columns.

The Colorado furnaces are built after current designs, with a heavy brick shaft above a single battery of cast-iron jackets in narrow sections surrounding the hearth. The brick shaft is surmounted by a brick chimney, leading the gases off above the charge-floor. None of the works, as already repeatedly stated, were originally provided with appliances for condensing or collecting the flue-dust. Though it is true that the operation does not make as much flue-dust as other kinds of blast-furnace smelting, it is still an error to neglect the saving of the amount that is made.

Recently both Leadville and Kokomo have erected flues or chambers for this purpose, but as the flue-dust is simply being put back into the same blast-furnace without agglomeration of any kind (beyond moistening with water), the real advantages of collecting the dust are vitiated.

COST OF A ONE-FURNACE PLANT AT BOULDER, MONTANA, 1892.

Excavation, material, and labor	$515.32
Smelter building, material, and labor of erecting	6,293.74
Assay office, " " "	1,593.14
Motive power and blast machinery, including labor, etc., of setting same	7,256.69
Hot-blast stove, material, and labor of constructing	8,645.70
Blast connections (blower to stove, stove to furnace), material, and labor of placing	1,707.56
Blast-furnace, 36"×96", material, and labor	6,128.82
Pumps and water supply	1,944.07
Tools for constructing and erecting	420 29
Tools for operating (smelting)	813.84
Scales, all kinds	1,358.77
Divers supplies, iron, steel, etc	2,594.34
Construction labor not included in above	5,957.21
All freights on materials, all kinds	5,000.00
Total cost	$49,728.89

The power used (steam) was 68 horse-power in all.

Amount of lumber in all, 196,000 feet b. m.

Amount of rock in all, 122 cubic yards (for foundations).

Number of common red brick in all, 81,000, of which 40,000 are in the three-compartment stove; none in the furnace.

Number of fire-brick, 18,100 in all, of which 14,000 are in the stove and 1,000 in the furnace.

	Pounds.
Weight of castings in stove	104,230
Weight of castings in furnace and paraphernalia belonging to it (slag-pots, forehearths, etc.)	21,600
Weight of wrought-iron and steel in stove	13,500
" " " " furnace	30,000
" wrought and cast-iron in blast connections	17,100
" blowing and sampling machinery	22,000
" motive-power and transmission (boilers, engine, shafting, clutches, pulleys, etc.)	50,000

The main items of cost at Kokomo and Leadville were f. o. b. cars in Denver, for material alone.

Iron work for 33" × 140" furnace, 59,000 pounds,—$3,700, not including paraphernalia of slag-pots, forehearths, etc.

Iron work for stove, 85,000 pounds, including connections to furnace—$3.400.

Number of red brick used: in stove, 32,000, in furnace, 21,000.
" " fire-brick " " 11,000, " 5,000.

Total cost of Kokomo works, about $35,000.

The recent reconstruction and enlargement of the Leadville plant for three furnaces has cost, all told, $200,000.

PRELIMINARY ESTIMATE OF COST OF A ONE-FURNACE "PYRITIC" SMELTER, 90 TO 100 TONS DAILY CAPACITY.

	Weight. Pounds.	Cost.
Excavation, according to site—$500 to		$1,000
Smelter building and bins, lumber, 200,000 feet at $8.00	800,000	1,600
Smelter building and bins, all labor on same		3,000
Office and laboratory building		500
Laboratory outfit	500	600
Engine and boilers (2), 70 horse-power complete	34,000	2,500
Feed-pump, injectors, etc		200
Engineer's supplies	1,000	100
Shafting, pulleys, etc	10,000	600
Setting boilers, engine and machinery		400
Red brick for engine and boilers, 24,000 at $7.00.	108,000	168
Fire " " " 2,000 at $40.00.	12,000	80
Cornish rolls, 20"×10", complete	6,000	400
Blake crusher, 7"×10", "	9,000	530
" " 4"×10", "	4,700	250
Sample grinder, complete	600	75
All belting	1,000	400
Root blower No. 7	16,000	1,700
Blacksmith and machinist outfit, hardware, etc	1,500	450
Carried forward	1,004,300	$14,553

Brought forward		1,004,300	$14,553
Iron used in construction		10,000	400
Steel used in construction		1,000	100
Hot blast stove, complete (iron and tiles)—			
Iron, etc.	$3,400	85,000	
Fire-brick, 14,000 at $40	560	84,000	
Red " 40,000 " $7	280	180,000	
All labor	1,000		
			5,240
Blast-pipes (cold and hot blast) complete		5,000	350
Furnace, 33"×140", iron parts complete	$3,700	59,000	
Fire-brick, 2,500 at $40	100	15,000	
Red " 10,000 " $7	70	45,000	
All labor on furnace	450		
			4,320
2 forehearths, complete		6,600	250
2 catch-pots, complete		1,500	80
12 slag-pots, complete		6,000	300
All scales, including 36 foot, 50-ton railroad scales and labor on same, weighing	6,000 3,200 1,600 500	11,300	1,300
Tools for operating (barrows, shovels, etc.)		2,000	300
Pump for water supply		1,500	350
Water supply tank, 29,000 galls., lumber and labor			300
All pipes, fittings, valves, etc		1,500	500
Lime, sand, and cement			150
Elevator, tram-cars, and tramway		5,000	500
Construction labor, not included above			5,000
Freight on material		1,523,700	variable
Total cost			$33,993

CHAPTER XVI.

REVERBERATORY FURNACES.

THIS method of smelting copper ores is peculiarly English, the reverberatory furnace having practically had its origin in Swansea, where it has, during the past two centuries, undergone various changes and improvements, by which its capacity and economy have been considerably increased without any radical alteration in its original form or practice.

This particular branch of metallurgy having engaged the attention of English and American authors to a greater extent than any other, nothing would be gained by a mere repetition of what may be found in the modern text-books, and the writer prefers to devote his own work to those practical details of construction and management that are yet wanting, and which he hopes may supplement the more strictly scientific information just referred to.

Within the past fifteen years, the conditions in the United States have been adapted to the use of this type of furnace for certain classes of ore, and the intense Western competition, combined with a new field for enterprise freed from the trammels of tradition, has resulted in not only more than trebling the capacity of the best Welsh reverberatories, but also in greatly simplifying and cheapening the entire process.

In economic improvements, it has outstripped the blast-furnace until the ratio between these two forms of apparatus has been so changed, that in cases where only ten years ago the blast-furnace would have been at once selected for some given set of conditions, it is quite possible that at this time the modern reverberatory would do the same work more economically.

THE CHEMISTRY OF REVERBERATORY SMELTING.

The final result of the fusion of sulphureted copper ores in either blast-furnace or reverberatory is essentially the same. In both cases, the properly conducted operation yields two products, aside from the gases that we need not consider. These are:

1. Slag, consisting mainly of the earthy portions of the ore, together with such of the iron present as became oxidized during the preparatory calcination of the ore (as well as during any further calcination it may have undergone during the process of fusion).

2. Matte, composed of subsulphide of copper, with such an amount of sulphides as can be formed from the surplus sulphur that remains after all the copper is first satisfied.

Gold and silver are also quite thoroughly collected in a copper matte, where they are chemically of but little moment; though they may be of vital importance commercially.

Lead, zinc, antimony, arsenic, tellurium, bismuth, platinum, nickel, cobalt, and several other substances collect in the matte to a greater or less extent, but need not be considered in this connection.

The principal reactions that take place in the laboratory of a reverberatory furnace, working on an ordinary mixture of partially calcined sulphides of copper and iron, containing sufficient silica to make a fusible slag, are as follows (or, at least, may be thus regarded for practical purposes):

1. The unoxidized sulphides melt, and reacting upon the oxides and sulphates ot copper and iron, are themselves partially decomposed, while the resulting sulphurous acid has a reducing effect upon both of the oxides of copper, as well as upon any ferric oxide that may be present, which latter is reduced to ferrous oxide; thus being left in a suitable condition to combine with silica to form slag.

$$Cu_2S + 2\,CuO = 4\,Cu + SO_2.$$
$$Cu_2S + 4\,CuSO_4 = 6\,CuO + 5\,SO_2.$$
$$Cu_2S + 6\,CuO = 4\,Cu_2O + SO_2.$$
$$Cu_2S + 2\,Cu_2O = 6\,Cu + SO_2.$$
$$2\,Cu + 2\,FeS = Cu_2S + 2\,Fe.$$
$$Fe_2O_3 + Fe + 3\,SiO_2 = 3\,FeO,\,SiO_2.$$

2. Oxides of copper, having a tendency to combine with silica, would enter the slag, were it not for the presence of sulphur. Thus:

$$Cu_2O + SiO_2 + FeS = Cu_2S + FeO,\,SiO_2.$$

The important reaction by which the sulphur reduces the infusible sesquioxide of iron to the slag-forming protoxide may be thus shown:

$$FeS + 3\ Fe_2O_3 + 7\ SiO_2 = 7\ FeO,\ SiO_2 + SO_2.$$

3. For practical purposes, the alkalies and alkaline earths may be assumed to combine directly with silica to form slag.

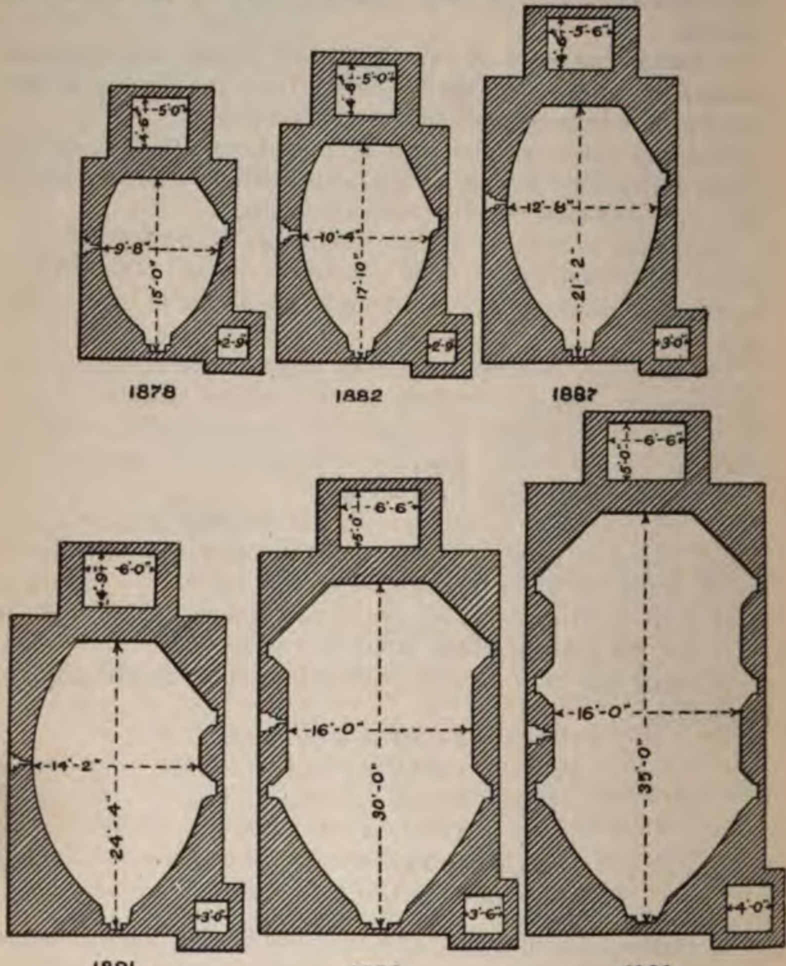


Fig. 52

The above reactions show three important points very clearly:

(*a*) Why a reverberatory furnace, with its neutral atmosphere, produces a richer matte from the same ore-charge than a blast-furnace, which has a powerfully reducing atmosphere.*

* We are now also using the blast furnace as an oxidizer.

(*b*) Why it is essential to have a sufficient amount of sulphur present in the charge; it thus being necessaryto add raw sulphides if the calcination of the ore has been carried too far.

(*c*) How important it is to avoid producing ferric oxide during the calcination process, in order to save the time and fuel required to reduce this sesquioxide back to a protoxide (ferrous oxide), and thus fit it for slag-formation.

The separation of the matte and slag depends mainly upon the difference in their specific weight. Hence, it is difficult to keep the copper low when basic, ferruginous slags are produced, while an earthy or siliceous slag is usually very free from grains of matte.

THE EVOLUTION OF THE MODERN REVERBERATORY.

It would be interesting to follow the development of the reverberatory furnace from the 9-foot by 14-foot hearth of former years, with its smelting capacity of 12 tons per 24 hours, to the 50-ton furnaces now in use, with (in one case) hearths of 16 feet by 35 feet in the clear. But we must content ourselves with the changes of the past 16 years, and I select as an illustration the Argo Works of Colorado, both because this is a reverberatory plant pure and simple, running on a variable mixture of unusually difficult ores, and also because its manager, Mr. Richard Pearce, has not only kindly supplied me with all necessary information, but has always kept accurate records from which this information can be drawn.

Fig. 52 represents the shape and size of the Argo reverberatory smelting furnaces from 1878 to 1894 inclusive, and the following table gives such further information as is necessary to appreciate the advantage that has been derived from this increase in size, and improvement in shape, of hearth.

TABLE SHOWING DEVELOPMENT OF ARGO REVERBERATORIES FROM 1878 TO 1894.

Year.	Dimensions in Feet.			Area in Square Feet.			Ratio Between		
	Stack.	Fire-Box.	Hearth.	Stack.	Fire-Box.	Hearth.	Stack.	Fire-Box.	Hearth.
1878......	2.75	4.5 ×5	9.66×15	7.56	22.5	105	1	2.98	13.9
1882......	2.75	4.5 ×5	10.83×17.83	7.56	22.5	189	1	2.98	18.4
1887......	3	4.5 ×5.5	12.67×21.17	9	24.75	198	1	2.75	22
1891......	3	4.75×6	14.17×24.83	9	28.5	265	1	3.17	29.5
1893......	3.5	5 ×6.5	16 ×30	12.25	32.5	402	1	2.65	32.8
1894......	4	5 ×6.5	16 ×35	16	32.5	481	1	2.03	30

Year.	Ratio Between		Per Cent Hearth Area Lost in Corners.	Ore Smelted per 24 Hours—Tons.		Coal per 24 Hours.			Tons Ore Smelted Per Ton Coal.	Pounds Ore Smelted per Sq. ft. Hearth per 24 Hours.
	Fire-Box.	Hearth.		Cold Ore.	Hot Ore.	Tons.	% Coal for Cold Ore.	% Coal for Hot Ore.		
1878.....	1	4.67	28	12	....	5	42	....	2.4	229
1882.....	1	6.18	24.5	17	....	7	41	....	2.43	245
1887.....	1	8	26	24	....	9	37.5	....	2.67	242
1891.....	1	9.3	23	28	....	10	36	....	2.8	211
1893.....	1	12.37	16	35	43	13	37	30	2.7–3.3	174–213
1894.....	1	14.8	14	....	50	13.5	....	27	3.7	208

It should be noted that the charges denominated "hot ore" consist of about 50 per cent. of calcined pyrites hot from the roasting furnaces and 50 per cent. of cold, unroasted siliceous and barytic ores.

The coal used in these furnaces has the following composition:

Water, at 100° Cent.	1.40	per cent.
Fixed carbon	54.90	"
Volatile matter	32.90	"
Ash	10.80	"
	100.00	"

The average smelting mixture is represented by the following analysis:

Silica	33.9	per cent.
Iron	10.8	"
Barium sulphate	15.5	"
Alumina	5.6	"
Carbonate of lime	8.5	"
Carbonate of magnesia	5.8	"
Zinc oxide	6.1	"
Copper	2.0	"
Sulphur	5.1	"
Oxygen	6.4	"
	99.7	"

The foregoing table presents many points that will appeal at once to the practical smelter. In the furnace of 1878, the area of the hearth was 4.67 times that of the fire-box, while the reverberatory of 1894 has a hearth area of 14.8 times that of the fire-box, the intermediate years augmenting regularly in the ratio of hearth to fire-box, as the size of the furnace increases. The same increase of ratio will be observed between the areas of hearth and stack, being only 13.9 to 1 in the smallest furnace, and increasing in the largest to 30 to 1. By referring to the cut, it will be noticed how greatly the shape of the smelting hearth has been changed in the newer furnaces, the tendency being to make it more and more rectangular and thus to avoid the loss of area due to the rounding off of the corners, especially at the front end. The loss in the furnace of 1878 by thus cutting off the corners amounted to 28 per cent., which has gone on decreasing until the furnace of 1894 shows a loss of only 14 per cent.; just one-half as much as in the older furnace. As is quite natural, the weight of ore smelted per square foot of hearth area has somewhat fallen off since the adoption of the rectangular shape, having reached the maximum amount of 245 pounds per square foot of hearth in the furnace of 1882, and decreased to 174 pounds in the furnace of 1893. The adoption of hot-ore charging brought the 1893 furnace up to 213 pounds per square foot, which again fell off to 208 pounds per square foot in the still larger and more rectangular furnace of 1894. But this is a very small matter compared with the saving made by the steady decrease in the consumption of coal per ton of ore smelted, which has accompanied the enlargements of the furnace. The smallest furnace smelted 2.4 tons ore with one ton coal. The furnace of 1893 smelted 2.7 tons cold ore, or 3.3 tons hot ore, and the great furnace of 1894 smelts 3.7 tons hot ore with one ton coal. These furnaces are shown in Figs. 53, 54, and 55.

The very slight increase in cost of plant necessitated by the smaller amount of ore smelted per square foot of hearth area is many times outweighed by the saving in fuel, in labor, in buildings, and in assays and all other work which is augmented by an increased number of furnaces. The furnace of 1893 offers an interesting opportunity to study the increase in smelting capacity of a furnace due to the use of hot ore. One-half of the "hot-ore" charges consists of hot calcines from the roasting furnaces, the other half being composed of cold siliceous and barytic ores. The reverberatory smelts 8 tons per 24 hours more of the hot than of the cold charge, a gain of 23 per cent. The cost for fuel per ton of cold ore smelted is about \$0.93, while the hot ore is smelted for \$0.76, a saving of \$0.17 per ton of ore, or some \$2,500 per annum, which represents, at 6 per cent. per annum, a sum of over \$40,000.

But it must not be lost sight of that this great increase in capacity and decrease in costs is not derived solely from the enlarging of the smelting furnaces. Merely constructing a large furnace does not necessarily imply the smelting of a large amount

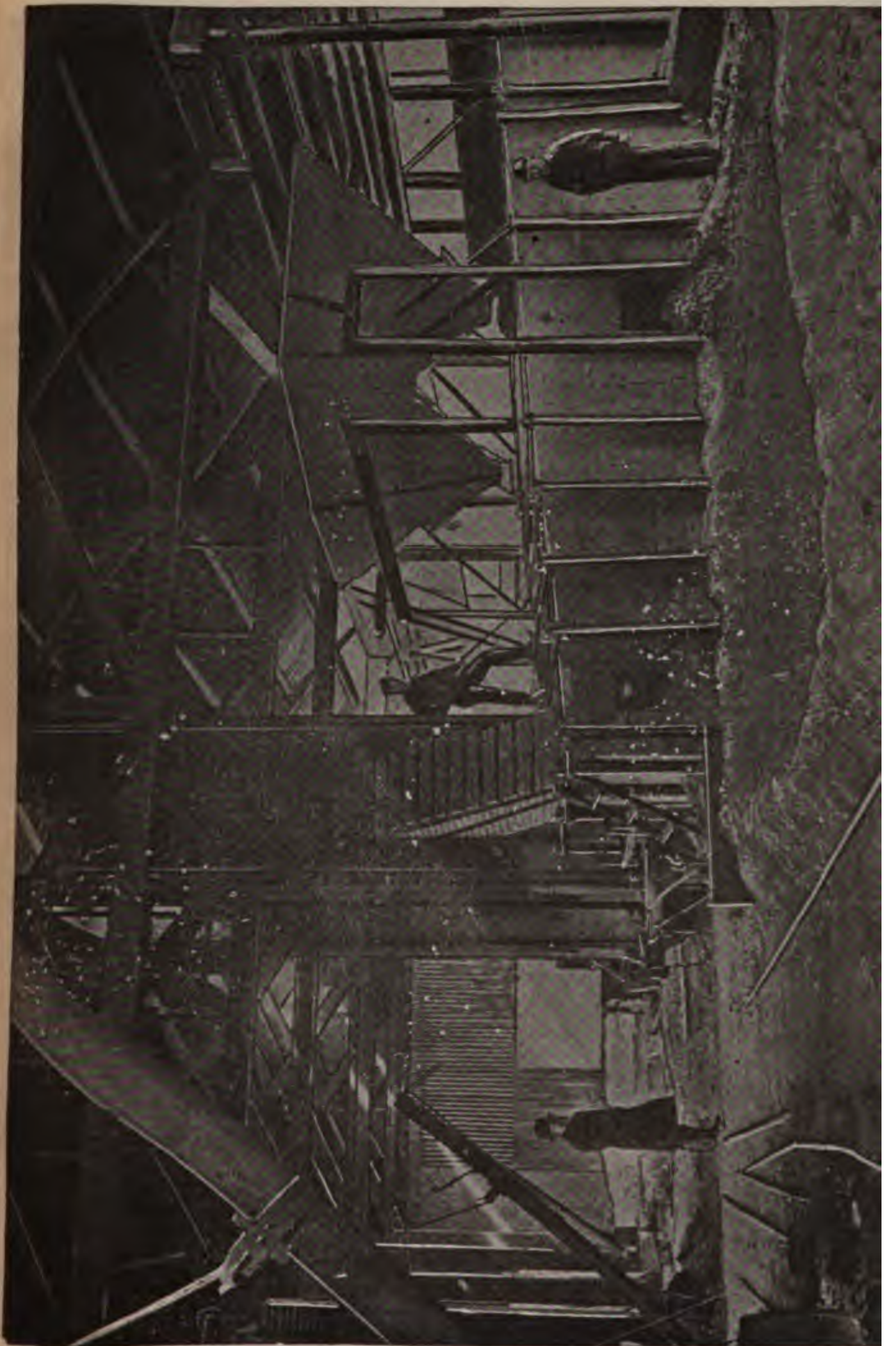


FIG. 55.—ARGO REVERBERATORY OF 1894.

of ore per 24 hours. A considerable proportion of the extraordinary results obtained at Argo are due to minor, but very important, improvements introduced by the manager, Mr. Richard Pearce. Some of these are not yet given to the public, but I may mention among them the careful and thorough preheating of the combustion air, by conducting it through channels in the body of the furnace; the excellent arrangements for charging; and, above all, the devices for rapidly skimming the vast quantity of slag resulting from a charge of 12½ tons of ore containing only 2 per cent. copper and producing a matte approaching 40 per cent. Some of these arrangements will be described and illustrated in this chapter.

The moral to be drawn from the success of these very large angularly-shaped hearths is manifold. Among other hints, it shows us that there is no difficulty in making large bottoms stand; that an arch with a 16-foot span is as easy to keep up as one of only 8 feet, providing it is properly supported at the skewbacks and allowance is made for its rising and settling with variations of temperature; that the hearth has not got to be made to conform to the supposed shape of the flame, but rather, that the flame (within reasonable bounds) will heat any decently-shaped hearth by radiation; and, what will appear the most extraordinary fact of all, to Welsh furnace labörers, that a reverberatory can actually be skimmed at the side as well as at the front.

Some of the main interesting features connected with the Argo furnaces are:

1. Rapid charging and spreading of the charge, the latter operation being greatly facilitated by the large body of matte in the hearth.

2. Preheating of the air by conduction through channels in the brick-work of the furnace.

3. Rapid skimming through four doors simultaneously, and conducting the molten slag in launders outside of the building, where it is loaded on railway cars, for ballast, at a profit to the smelting company.

4. Tapping only weekly, say after 24 or more charges, thus keeping the hearth well covered with matte.

The practice at Butte, Montana, is of quite a different character, the charge there consisting mainly of calcined, pyritious concentrates, carrying 10 per cent. to 22 per cent. (usually about 15 per cent.) copper, to which are often added sufficient dry, siliceous silver ores, to make a slag containing 30 per cent. to 45 per cent. silica.

Some of the reverberatory work at Butte is hampered by an injudicious method that prevails there of handling the flue-dust from the blast-furnaces. This unwelcome substance is made in large amounts in cupolas run on a finely divided charge, with low ore-column and strong blast. The resulting flue-dust not only contains an undue proportion of zinc, but also a considerable amount of ferric oxide, one of the most stubborn and disagreeable substances that can be charged into a reverberatory furnace.

As a matte of above 50 per cent. copper is ordinarily aimed at, it follows that the concentration is only about 3½ (at some works 2½) into 1, the matte fall thus being nearly 30 per cent. of the total weight of the charge. This makes a very favorable condition of things for rapid and easy fusion, and to this is also added the fact that 50 per cent. to 80 per cent. of the entire charge often consists of hot calcines.

But even this is scarcely sufficient to explain the unprecedented capacity attained by the reverberatories of certain Butte smelters, such as those of the Montana Ore Purchasing Company, and the Butte & Boston Mining Company; an average of 50 tons per 24 hours being sometimes maintained for many successive days, and some 48 tons per day for a run of months.

I have late advices from a Butte metallurgist of undoubted reliability and accuracy, stating that for several successive days a new reverberatory at one of the works above mentioned, with hearth 13 feet 9 inches by 22 feet in the clear, has averaged 73 tons (146,000 pounds) per 24 hours, on a mixture of hot, calcined pyritic concentrates and siliceous ores.

The principal features of the Butte practice, that determine this enormous capacity of very moderate-sized furnaces, are:

1. A more rapid combustion of fuel (say 10 tons per day), the capacity of the furnace, however, increasing in a more rapid ratio than the consumption of fuel. This rapid burning of coal is effected by:

(*a*) Enlarging the flue and stack areas considerably beyond what has been considered their normal proportions. (Increasing the height of the stack beyond 60 or 65 feet does no good here, nor have I ever found it of any advantage in any reverberatory work, except where there were hills, or higher buildings in the immediate vicinity.)

(*b*) Slightly enlarging the grate area, and keeping the live coals very close to the grate-bars, thus totally abolishing the clinker grate, which was introduced to allow the use of a certain proportion of anthracite fines, at Swansea.

2. Endeavoring to keep the bottom of the furnace hot, instead of trying to cool it from the vault. A very large body of matte, 6 to 15 tons or more, is constantly kept on the hearth; and as the men are prevented, by an iron plate, from tapping too low, the bottom does not wear down unreasonably. Some metallurgists dread the effect on the hearth of this large body of hot matte. Its effect is in reality a protective one. It is true that, in smelting blister copper, its boiling, its high conductive power, and great weight tend to make it a somewhat severe substance on a sand-hearth. But a large body of 50 per cent. matte is a very thorough protector for a properly constructed hearth, interposing an effective shield to the intensity of the flame, and, as a neutral substance, preventing the corrosive action of the ferrous oxide in the charge. With a large body of matte constantly in the furnace, I have found less wear on a bottom after a year's run than had formerly occurred in a month where the bottom was left naked every charge or two.

3. Lessening the intervals between charges, and also lessening the cooling off of the furnace. This is accomplished by rapid skimming and dropping the fresh charge direct from hoppers upon the pool of molten matte in the furnace, which thus acts to disperse the hot calcines, which indeed are ready of themselves to spread in every direction like a semi-liquid substance.

It is hardly an exaggeration to say that from 30 per cent. to 50 per cent. of the time and fuel consumed in running reverberatory furnaces on this principle was wasted in two directions that were totally useless and unnecessary. These were, and in many works still are:

1. In reheating the furnace after it has been unnecessarily cooled by:

(*a*) Slow charging through the side doors, instead of from hoppers.

(*b*) Slow leveling of the charge, from failing to keep a large body of molten matte in the furnace to assist in spreading the fresh charge.

(*c*) Careless grating, by which large holes are left in the fire.

(*d*) By allowing the calcined ore to cool before it is charged into the reverberatory.

(*e*) By omitting to provide a large opening in the side of the stack, above the flue, which can be opened whenever the furnace doors are taken down, thus checking the rush of cold air through the hearth, and confining the cooling to the stack, where this action is desirable.

(*f*) By too frequent fettling, and by wasting too much time over this operation. Furnaces running with moderately siliceous slag should not require fettling oftener than every 5 days. A basic slag will cut a groove under the side-lining and the bridge-wall, and when this groove has become deep enough to demand attention, it should be rammed full of a mixture of 80 per cent. quartz (hazelnut size), and 20 per cent. raw fire-clay. This should be thoroughly pugged together, and made into large balls ready for use, before the furnace doors are taken down. Two sets of men can clay the furnace simultaneously from the two opposite side doors; the one at the east door operating on the northwest section of the lining, while his partner at the west door is claying the southeast section, and vice versa. After this groove is once thoroughly filled, it can be kept from too rapid cutting by frequently throwing a heavy ridge of siliceous ore against it, and all around the edges of the hearth.

2. In waiting for the charge to be perfectly smelted before skimming off a portion of the slag; and in prolonged firing to "raise the charge off the bottom."

It is proper and advantageous to skim the furnace as soon as enough of the charge is fused to give a tolerable bed of clean slag, and then the rest will "come" in much less time.

As regards the charge sticking to the bottom, there need be little danger of this annoying impediment, if a large body of matte is always kept in the furnace. But where a sticky, slimy deposit has accumulated on the hearth and is gradually increasing, the quickest and cheapest remedy is a large charge of pyrites. The resulting great body of low-grade matte will clean out the hearth more quickly and thoroughly, and do the furnace much less harm, than attempting to remove it by the sole agency of long-continued heat.

The Anaconda Mining Company has constructed a reverberatory furnace that presents certain interesting features. Through the courtesy of officials of this company, I am enabled to present

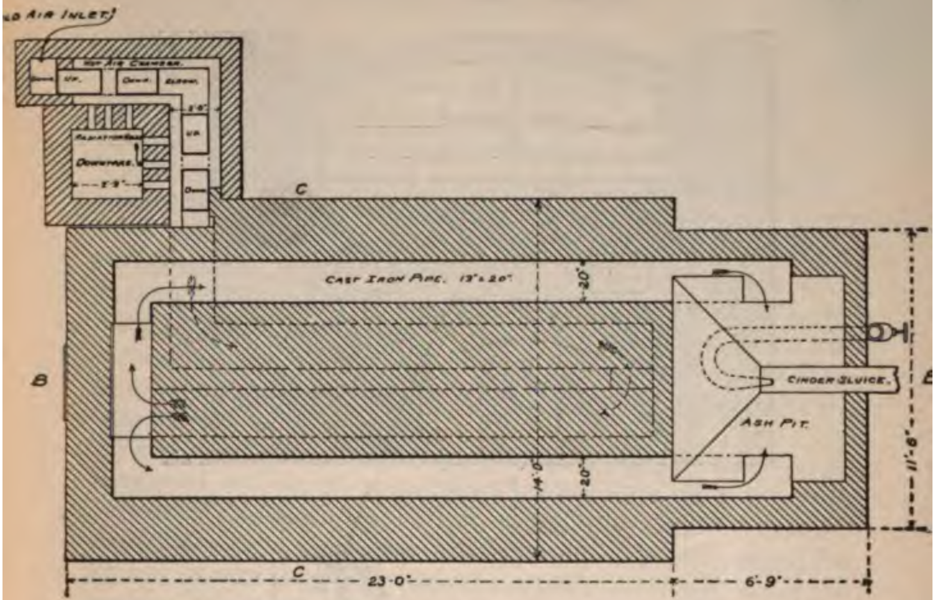
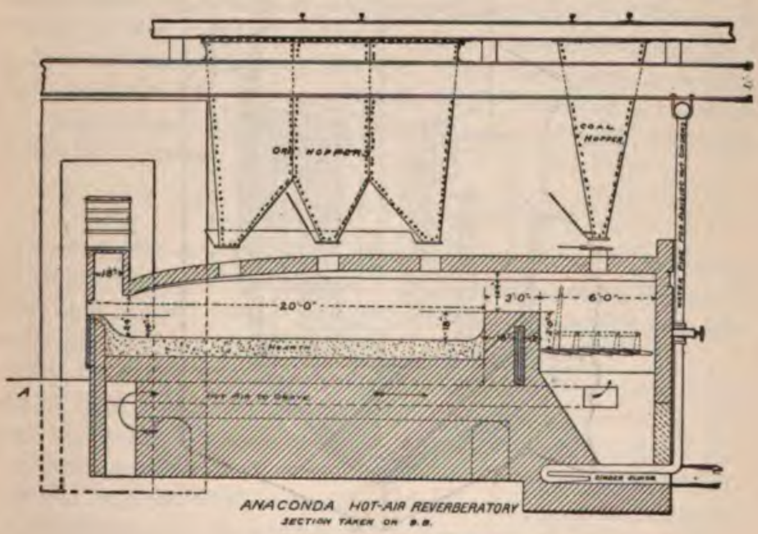


Fig. 56.



ANACONDA HOT-AIR REVERBERATORY
SECTION TAKEN ON B.B.

Fig. 57.
Anaconda Reverberatory Furnace.

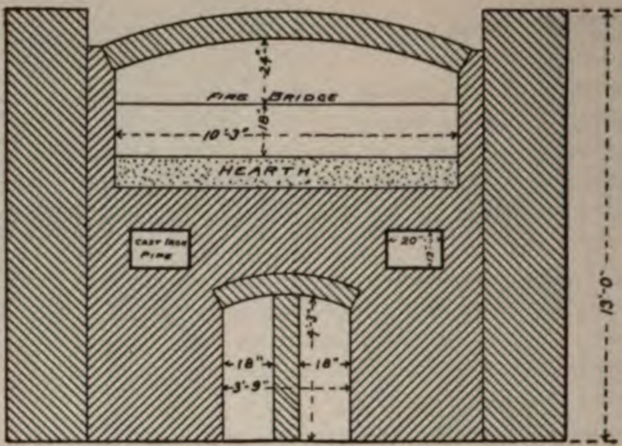


FIG. 58.

FIG. 59.

ANACONDA REVERBERATORY FURNACE.

drawings of this furnace, which is designed to use an inferior coal, containing 20 per cent. or more ash. (See Figs. 56, 57, 58,and 59). The coal is fed from a hopper direct into the fire-box, and the ashes are sluiced out of the hopper-shaped ash-pit by a current of water. The grates are shaken by an outside handle, like the grates of a pyrites burner.

As will be seen from the cuts, the most interesting feature of this furnace is its unique and elaborate arrangement for preheating the air used for combustion. Two sides of the stack, for a distance of 15 feet above, and 6 feet below, the ground line, are enclosed by outer walls, an air-tight space being left between. Some radiation holes are also left in the stack-wall, to more effectually heat this air-chamber.

This hot-air space is traversed by a zigzag pipe, that receives its current of outside air at the point indicated in the cut. After traversing this pipe, the hot air passes backward and forward through the main vault under the furnace, and finally reaches the fire-box by traversing large cast-iron conduits on either side of the hearth. The air is reported to be heated to a temperature of 700 degrees Fahr. (371 degrees C.) before it attains the ash pit.

Although the hearth is small, 10 feet 3 inches by 20 feet, the furnace is said to average 46 tons (92,000 pounds) of hot Anaconda calcines per day. If this be the case, a very great advantage must be derived from preheating the air. How much greater results might be derived from preheating the air by means of extraneous fuel, rather than by abstracting caloric from the hearth, would be interesting to know.

THE BUTTE REVERBERATORIES.

These are not so long as the Argo furnaces, 20 to 22 feet being the ordinary internal measurement of the hearth, while the width varies from 11 to 14 feet. But here we have unusually favorable conditions for rapid smelting.

Aside from the slag that goes back to the furnace (and is not counted in the weight of the charge), and a slight amount of siliceous silver or gold ores, almost the entire charge consists of calcined concentrates or of selected first-class ore; the two latter classes of material containing 15 to 22 per cent. copper, and being calcined down only to some 9 or 10 per cent. of sulphur. The desired concentration is only 3 or 4 into 1, as the matte produced in general is not expected to run higher than from 50 to 55 per cent. copper.

Such a charge gives not only a very fusible, ferruginous slag with high heat-conducting qualities, but also some 30 per cent. of a very fusible matte, that melts at the first reasonable heat, and forcing its way downward, soon lifts the charge from the hearth, and places it between two sources of heat.

The calcined ore is also commonly charged in a red-hot condition; thus, not only effecting an important saving in time and fuel, but also materially facilitating the spreading of the fresh charge in the furnace, there being so little cohesion between the matte-particles at this temperature that it flows almost like water.

Overlooking the slight differences that exist at the various smelters in Montana and Colorado, let us examine a fair type of modern reverberatory furnace, and note the practice and results.

MODERN REVERBERATORIES.

The first point to determine, in beginning the construction of a new reverberatory, is, whether the furnace shall be erected over a cooling vault in the customary manner, or be built upon a solid foundation.

I suppose that we cool the under side of a reverberatory hearth because it has always been the custom to do so. But when one reflects that the hearth of a reverberatory is the part that is always too cold, being invariably covered with a layer of poor conducting material; that consequently the ore-charge always sticks to the bottom until the very last; and that frequently it takes nearly as much time and fuel to raise this crust from the bottom as it does to smelt all the rest of the charge, it would seem unnecessary to make matters worse in this direction than they already are.

I have long found that time could be saved in smelting a charge by closing up all openings to the vault, and thus keeping the bottom hotter. But Mr. C. M. Allen, of the Butte & Boston Smelting Works, has gone much further, and has skimmed some of his furnace vaults full of slag. Thus far, no symptoms of any undue deepening of the hearth have been noticed. When thus treated, it will deepen to a certain extent; but will soon arrive at a point where it remains stationary, providing it is kept well blanketed with a sufficient layer of matte.

The main cause of the too rapid deepening of a good bottom is the desire of each furnace-man to turn out a larger bed of matte than his partner on the preceding shift was able to do. Consequently, it is the common practice of these workmen to drain the hearth as dry as possible, tapping lower and lower each succeeding time, and thus, not only cutting the bottom deeper mechanically, but leaving it fully exposed to the strong corroding action of the oxides contained in the next ore-charge. This may be effectually prevented by placing an iron plate under the tap-hole, and insisting that the furnace shall never be tapped lower than at a fixed point, which will always leave several inches of matte to protect the bottom. I shall return to this important point, when speaking of the general management of reverberatory furnaces.

It is quite possible that it will be found best to build our future reverberatories on a solid foundation, entirely omitting the vault, but substituting a thick layer of carefully tamped steep or clay.

The general construction of these large reverberatories is practically the same as for the smaller ones hitherto in use; due attention being paid to their general strength, and to the necessity of much stronger anchoring and tying.

In modern furnaces, the pear-shaped hearth is pretty much abolished, nearly the full width being maintained for the greater portion of its extent, until it is quite suddenly contracted toward the skimming-door.

The capacity is thus considerably augmented, while the volume and radiation of the heated gases seem quite sufficient to furnish a sharp smelting heat to the broadened hearth.

A large furnace, that is to do the most rapid smelting possible, should have a stack at least 48 inches square and 65 feet high. The flue should be 24 inches wide, and also 24 inches high (measuring at right angles to its slope), at the furnace end; and 36 inches high where it enters the stack. If it is found too large, it can easily be dammed.

The space between arch and bridge-wall, as well as the area of the grate, are two factors that depend so greatly upon the quality of the coal used, as well as upon other local conditions, that it is impossible to give any exact figures regarding them. They should be built amply large in proportion to the hearth and flue-area, and can easily be contracted if it is found that too much fuel is consumed.

It is not at the initial point of combustion that a great space is required; but it is immediately after the full oxidation of the gases has occurred (assisted by air introduced through the arch, over the bridge), and when they have attained their maximum temperature, that ample room is needed for their enormously increased bulk. The sudden enlargement of the reverberatory furnace from the combustion chamber to the hearth is admirably adapted to the requirements of the case.

For moderately good coal, with 8 or 10 per cent. ash, the fire-box might be made 4 feet 3 inches by 5 feet 6 inches inside, the space between bridge and roof being, of course, of the same length as the fire-box, and perhaps 28 inches high in the center.

Mr. R. Pearce has, for many years, made in his furnaces a certain modification of the arch directly above the bridge-wall, which serves to suddenly depress the flame, and which produces an effect upon the driving of the furnace entirely out of proportion to its apparent insignificance. I believe that its advantages have always been noticed in the various works that have made use of it. The following cuts illustrate a modern reverberatory, and show the sudden depression of the small portion of the main arch immediately above the fire-bridge, and give a general idea of its extent and proportions, which may be varied to suit conditions.

The distance between the grate and the upper surface of bridge-wall is of much importance, and also generally serves to determine the depth of the fuel-bed, as the ordinary fireman shovels in coal till it is about level with the surface of bridge.

As most of the reverberatories in the United States employ a bituminous, or semi-bituminous coal, the deep clinker-grates of the Welsh smelters have been generally discarded. Experience has shown that for our conditions, where fast driving of the furnace prevails, even if accompanied with some waste of fuel, a comparatively thin layer of fuel, with live coals nearly down to the grate-bars, gives the best results.

Where fuel is expensive, the pieces of coked coal that drop through the grates at the frequent barrings, may be cheaply recovered by washing, and used under the boilers by means of a suitable mechanical stoker.

Twenty-four inches is a common measurement at present for the distance between grate-bars and bridge-surface; but I have seen the most rapid smelting in the United States done with only 19 inches of depth, and with coal containing 10 per cent. ash.

In the tying of the furnace, I-beams will generally be found more economical than rails, as the same strength can be obtained with a saving of 50 per cent. or more in weight.

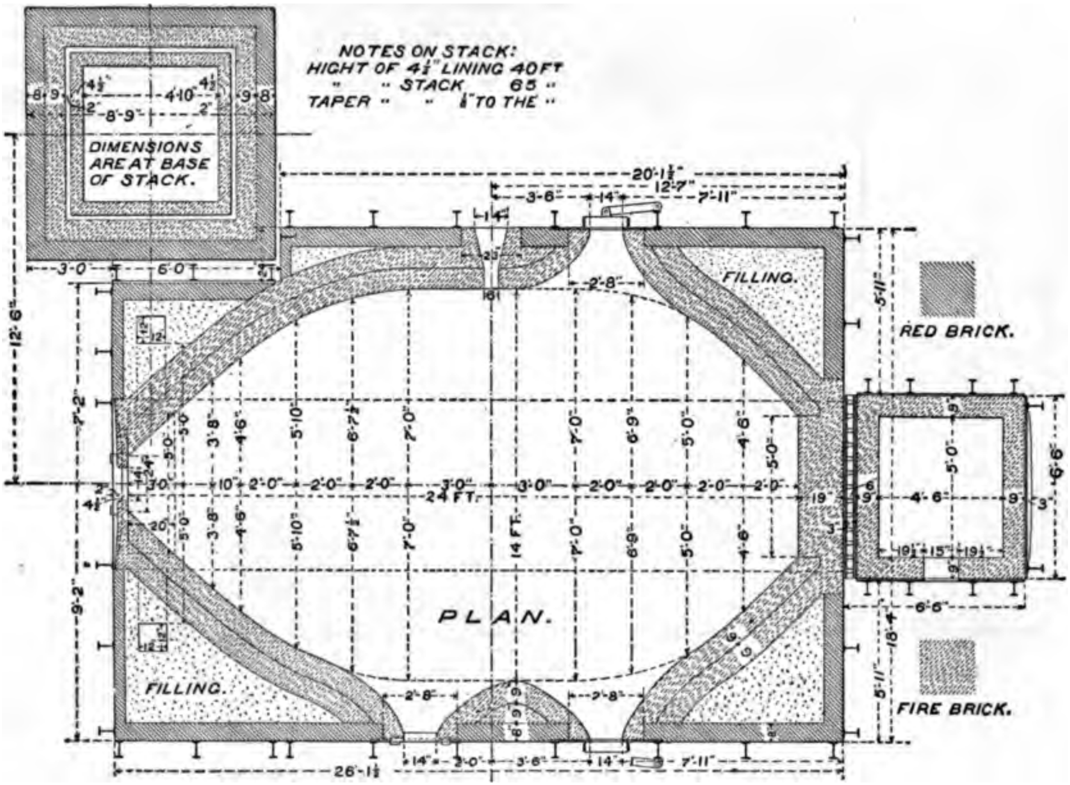
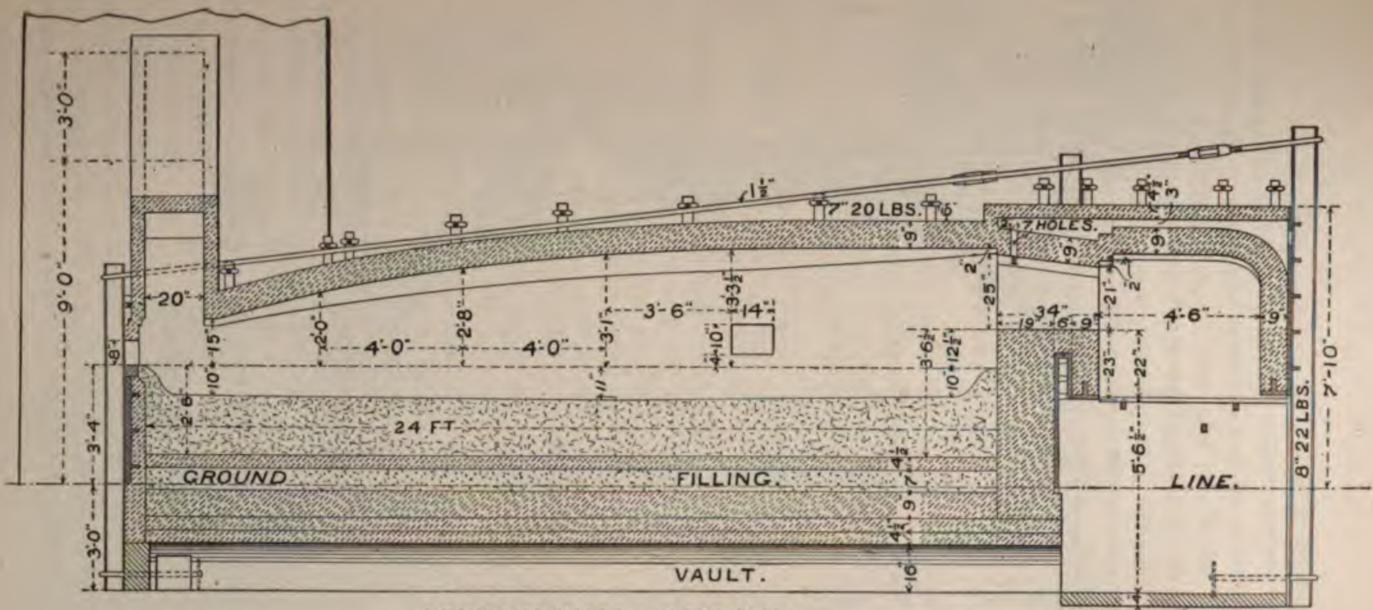


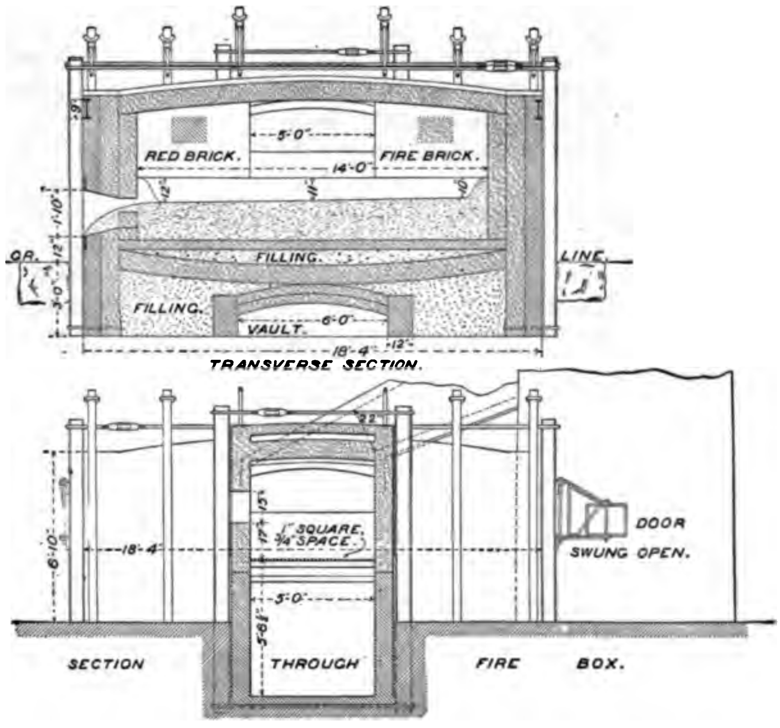
Fig. 69.—Modern Reverberatory Furnace.



LONGITUDINAL SECTION.

FIG. 61.

Where the vault under the hearth is omitted, difficulty may be experienced in tying the lower ends of the cross-buckstaves. With these very large furnaces, it is scarcely safe to depend upon a short cross-rod, with a turned down hook buried in the foundation of the furnace. Wherever it is practicable, it is well to pour molten slag from another furnace all around the lower ends of the buckstaves, even continuing the same to form the flooring of the



FIGS. 62 AND 63.

smelter-house; always providing that the slag is not so basic as to crumble too easily, or does not contain so much sulphide of calcium as to greatly deteriorate by slaking. An otherwise unsuitable slag may sometimes be rendered fit by covering it with a thick layer of sand, and thus annealing it till cold.

The I-beams are often placed in pairs, a single 2-inch tie-rod answering for each pair. I-beams are also to support the skewbacks around the entire furnace. The Parrot smelter at Butte uses special Dinas bricks for its reverberatory arches, 12 inches in length, and proportionately as large in their other dimensions. They claim that they obtain a decided advantage from the increased life of the arch.

In general, the ordinary Dinas brick are used, both for arch and inside lining-walls.

The increased size of hearth demands an increased number of openings for spreading the charge and fettling the hearth conveniently.

There is a decided advantage in having two doors on each side of the hearth, and in very large furnaces there is an extra skimming-door on one or both sides, the front door being entirely inadequate to the removal of some 5 to 8 tons of slag in a few moments.

All modern reverberatories are supplied with rational doors, mounted on strong iron hinges, that are either built into the brick-work of the furnace, or fastened to a neighboring column or buckstaff. The doors are made larger than the respective openings into the furnace, so that the iron frame of the former is not exposed directly to the heat.

The ore is almost invariably fed into the furnace by means of hoppers. These should be of heavy boiler-iron, and firmly supported by independent iron columns and I-beams. There is a difficulty about making a proper connection between the delivery end of the hopper and the hole in the arch of the furnace. Mr. Pearce has overcome this at Argo by the use of an ingeniously planned funnel of iron that is raised to form the connection between hopper and hole in arch.

To distribute the heavy charges over the large hearth now used, it is found best to employ two to three hoppers, situated in the long axis of the furnace. At Argo, with its 35-foot hearths and 12½-ton charges of ore, three hoppers are used.

The tracks from the ore-bins, and when possible to use hot ore, direct from the calcining furnaces, are run immediately over the hoppers, and the handling of the charge is thus reduced to a minimum.

The means of handling the slag and matte will be referred to when discussing the management of the furnace.

The coal should be brought to the reverberatories on an overhead track, and where feasible, dumped from the railway trucks direct into the furnace-bins without further handling. The lay of the ground will, however, seldom permit this, and it is not well to sacrifice too much therefor.

At the Anaconda works, and with the use of a forced blast, the coal is dumped directly into a hopper above the fire-box arch, and the furnace thus fired automatically; the grating being accomplished by means of hinged and pivoted grate-bars, that are moved with a lever as in many roast-kilns. The ashes fall into a conduit, toward which they are directed by the sloping sides of the ash-pit, and are removed by a stream of water. (See Fig. 57.)

This innovation is reported to work well, but I cannot speak from personal trials of it. It is arranged for very poor coal with some 20 per cent. ash, that is not inclined to clinker badly. Under such circumstances, I cannot see why it should not be advantageous.

We still are hampered by a certain superstitious veneration for many old, tedious, and expensive operations in connection with reverberatory smelting. They are becoming rapidly dispelled in American practice, and the sooner we can get rid of the disagreeable and expensive practice of hand-grating, the more profitable and pleasant it will be for both workman and master.

THE REVERBERATORY HEARTH OR BOTTOM.

The actual fire-brick box that is to contain the future hearth is usually bounded by the side-walls of the furnace laterally, by the bridge-wall at the rear end, and by the front wall at the skimming-door end. The bottom proper consists of an inverted arch of fire-brick, the lower ends of which, at their deepest or most concave point, are nearly in contact with the upper ends of the fire-brick that form the upper little arch of the cooling-vault. When this vault is omitted, a bed of steep, some 4 inches thick in the center, should be laid down in shape to receive the inverted arch that is to form the proper bottom of the furnace. It seems quite possible that in the course of time we shall follow the example of certain English works and smelt directly upon this brick bottom, in which case it must be placed considerably higher than when we intend following the customary practice of using sand-bottoms.

It was formerly the practice to smelt in at least two bottoms; but owing to the frequent wearing through and rising in patches, of the upper layer, many metallurgists now prefer to make but a single bottom of medium thickness. The preparation and smelting-in of these sand-bottoms is described elsewhere.

BRICK BOTTOMS.

Owing to the rapid wear of sand-bottoms, and more especially to their unforunate capacity for absorbing large amounts of the valuable metals, various metallurgists have tried smelting directly upon an unprotected brick bottom.

The brick bottoms, as constructed at the Burry Port Works and at the Messrs. Elliott's Metal Company's Works, both in South Wales, seem to me the most durable and satisfactory that I have ever seen.*

At the Burry Port Works (Cape Copper Company), no other kind of bottom is used, be it for smelting, blister, or refining-furnace. I carefully examined a bottom that had had two years of constant use, and could find no evidence of wear; only the surface of the bricks being glazed and crusted. Only a few hundred pounds of copper are needed to saturate such a bottom, and absorption of metal then ceases, there being indeed no place into which it can escape.

At the Elliott's Metal Company's Works, where 17 such bottoms have been long in use, very refractory silver ores are treated, containing much blende, and other objectionable substances, and requiring a prolonged high temperature in the furnace. I am informed by the manager, that one of these brick-bottoms has outlasted 10 sand-bottoms and is still in use.

Although a patented invention, a description of the manner in which these bottoms are constructed will be interesting to many. I am indebted to Mr. James for information and plans.

The furnace bottom consists merely of an inverted arch of brick, 9 inches, or even 4½ inches in thickness.

Dinas brick are sometimes used; but owing to their tendency to expand, Stourbridge brick are usually preferred.

This inverted arch is laid upon a bed of carefully rammed brasque (steep), 6 inches thick in the center and increasing toward the sides and ends, in order to give the proper curve to the hearth, as well as the necessary inclination toward the tap-hole; or, if a refining-furnace, toward the ladle-hole in front.

This brasque is made in the common manner, out of two portions of rather lean, raw clay, and one of coke-dust, and after being slightly dampened and thoroughly mixed, is carefully rammed into place, and shaped to receive the brick.

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* Patented by Messrs. Christopher James and William Griffiths, of South Wales.

These are put in as close as possible, and well grouted with a lute made from pulverizing some of the bats. Of course, the arch must be well keyed, and must take a proper bearing all around at the sides, as it is mainly its arched form that prevents its being floated up when a charge of 15 or 20 tons of molten copper is resting on it.

After drying, the furnace is all ready to start, without any waste of time or fuel in smelting-in bottoms.

Figs. 64, 65 and 66 are too plain to require explanation.

HOT AIR IN REVERBERATORY PRACTICE.

Most of the modern reverberatories are provided with some device for heating the air that passes through the apertures in the arch above the bridge. Some of them go so far as to heat the air that passes under the grate.

There is, no doubt, considerable gain in heating the air that is to enter the furnace or grate, if it can be brought to a sufficiently high temperature, and if the devices for its heating do not tend either to cool the furnace, or to complicate or weaken its construction.

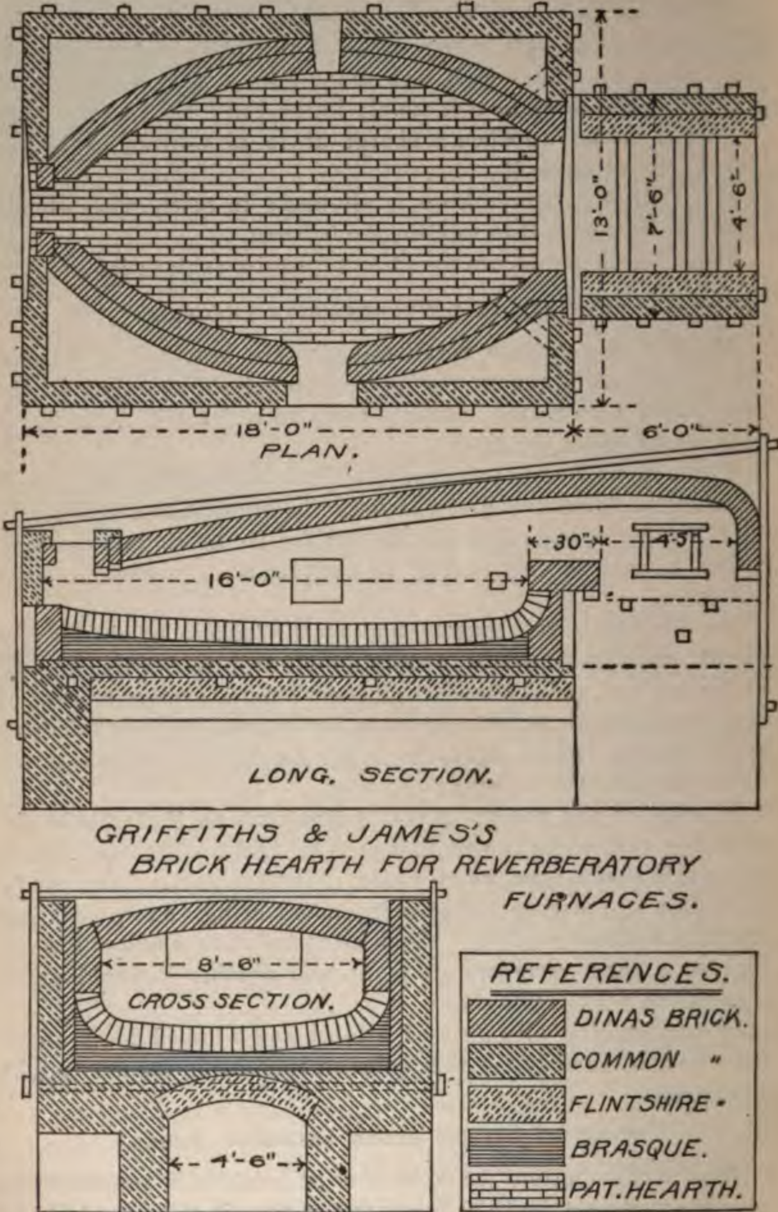
To complicate the construction of the furnace by air-passages parallel with its side-walls, or by a double arch or double end-wall at the fire-box, seems a most cumbersome and expensive method of heating a little air to 200 degrees or 250 degrees Fahr. (93 degrees to 121 degrees Cent.), the latter being the highest temperature I have ever obtained in testing the same.

In view of the greatly increased temperatures that can be obtained by preheating the air that is to be used in the combustion of coal, it would seem worth while, if it were worth doing at all, to employ a proper stove with independent firing, as in preheating the blast for iron smelting. Such an installation would only be in place at a large reverberatory plant, and whether it would prove economical must yet be shown.

In combination with a closed ash-pit and a forced blast under the grate, it seems to me that quite an increase in capacity might be expected.

DEPTH OF BOTTOM BELOW SKIMMING DOORS.

Whether the bottom be of sand or of brick, it is important to give it a sufficient depth at the outset to contain the large body of molten metal that is essential to the rapid smelting of heavy ore-charges.



FIGS. 64, 65 AND 66.

Many furnace-men have an idea that they are endangering their hearth by allowing a heavy body of molten matte to lie on it for days and weeks together. (Only being tapped when it is necessary to clay the furnace.) In this they are quite mistaken. Temperature is relative, and as hot as molten matte may seem to the eye, it is cool and safe, compared with the fierce heat of the flame on the naked hearth, or even of the long-continued high temperature that is frequently employed to raise crusts from the bottom.

Besides, matte, where it is not exposed to oxidation, is a neutral substance, having no chemical or corrosive action on the bottom, as do the basic oxides contained in a fresh charge of ore, that comes in direct contact with the sand-hearth.

Again, we have the cooling action of the fresh ore-charge every 3 or 4 hours; which, falling into the liquid matte, quickly chills it. The matte, being an excellent conductor of heat, thus rapidly cools the bottom, so that, on the whole, there is far less danger of trouble and corrosion when carrying a constant and heavy body of matte in the furnace than when tapping dry several times in the 24 hours.

(I need hardly say that I am not referring to metal such as "blister," that "works" violently. This fierce ebullition, accompanied by the transference of the surface heat to the bottom, through the excellent conducting medium of metallic copper, may cause much damage to a thin or patched bottom, by its mechanical violence.)

The most advantageous depth of the hearth will vary according to local conditions, and is especially influenced by the quantity of matte produced in proportion to the ore-charge, *i.e.*, the rate of concentration.

With a properly-ironed furnace with brick bottom, and for ores making 15 per cent. to 30 per cent. of their weight in matte, I should construct my bottom so that at the deepest point (the tap-hole side), it was some 16 or 18 inches below the skim-plate. I would keep my tap-hole opening some 8 inches or more above this deepest point, and thus always retain a body of 6 to 15 tons of matte in the furnace. A small, separate orifice, lower down in the tap-hole plate, will permit the draining of the furnace on the rare occasions when it is necessary to bare the bottom entirely in order to fettle the sides and joint more thoroughly than usual, or when a different class of material is to be smelted.

The advantages of such practice can best be appreciated after they have been tried. The fresh charge, which must be perfectly dry, and, if possible, hot from the calciners, is allowed to flow from the hoppers directly into the bath of molten matte. Falling from two or three separate openings in the arch, it floats for a considerable time upon the surface of the bath, and is often spread widely over the furnace before it begins to heap up unduly in cones under the hoppers. The workmen aid it with their paddles, and in five minutes a charge of as many tons of ore is spread, and the doors closed and luted. The men are saved half an hour of the hottest and most exhausting labor that can well be imagined, and the large amount of heat stored up in the great body of molten metal is rapidly absorbed by the fresh charge. A very short period of firing serves to restore the bath of matte to its former fluid condition, and in this we have a most potent agent to float up any unmelted masses that may be sticking to the bottom, and to place them in just the position where they must encounter the fierce heat of the naked flame.

The furnaces of the Butte & Boston Smelter, as well as of the Montana Ore Purchasing Company at Butte, now work off a 5-ton charge of ore, exclusive of any slag added, in 2½ hours. Having to handle large quantities of molten material so quickly, it becomes a matter of great importance to do so as rapidly and economically as possible.

A new reverberatory at the latter works, which is claimed to have a capacity of some 70 tons per day on hot calcines and siliceous ores mixed, has the following dimensions. Some of the measurements of the older 50-ton reverberatories are also given.

	50-ton Furnace		70-ton Furnace.	
	Feet.	Inches.	Feet.	Inches.
Outside length of hearth	22		25	4
Outside width	15		17	2
Inside length of hearth	20	6	22	8
Inside width of hearth	12	10	14	6
Fire-box (inside)			6	6 × 6 ft.
Width of bridge			2	10
Top of bridge to spring of arch			1	5
Top of bridge to center of arch			1	10
Width of flue			1	11
Height of flue at neck			2	7
Height of flue-opening into stack			4	6
Size of stack inside (square)			4	
Spring of main roof			1	2¼

HANDLING SLAG AND MATTE.

It is best to skim from a side-door, as well as from the usual opening in front. The arrangements of troughs and settling-pots used at Argo, to remove the slag, is shown in Figs. 53, 54, and 67.

Where water and fall are obtainable, it is usually most economical to granulate and remove the slag by this means. If sufficient fall cannot be obtained to wash the slag directly over the dump, and if water be scarce, a simple elevator can raise the granules to an artificial dump, while much of the water can be recovered. No particular mechanical device is necessary for the granulation of the slag. So long as there is a sufficient flow of water to break up the slag-stream as it falls into the water, and to prevent the formation of masses that are still liquid inside, the slag is easily granulated and carried away, making little steam, and requiring little attention.

At the Parrot Smelter in Butte, a 6-inch pipe with 12-foot head, and about 100 feet in length, supplies enough water to remove the slag from the reverberatory furnaces, of course only one being skimmed at a time. But it is found necessary to keep a man at the point where the water and slag meet, to break up any masses that may tend to form.

In comparing this practice with blast-furnace work, where the slag is granulated in the same manner, it must be borne in mind that 5 tons of slag are often skimmed from a reverberatory in 15 minutes, forming a flow of molten material that would correspond to a blast-furnace that was perhaps making 400 tons of slag per 24 hours.

Between the periods of skimming, the reverberatory requires no water at all. Hence, where water is scarce, it is well to store it in a reservoir or tank, large enough to hold the supply required for one or more skimmings, and allow it to fill up in the intervals.

Where the slag cannot be removed by this cheap and satisfactory method, it is often allowed to flow directly into the large Nasmyth pots, which, being mounted in pairs on trucks, are pulled to the edge of the dump by a mule, and there emptied in the customary manner.

Where the granulated slag can be used for ballast or filling, the launder may be allowed to discharge directly into the railway trucks, the water running to waste, while the granules remain behind.

Or, as at Argo, where the railway companies are glad to remove the slag, in order to use it as ballast, it may be run in clay-lined iron launders outside of the building, and cast into pigs in sand-beds. (See Fig. 67.)

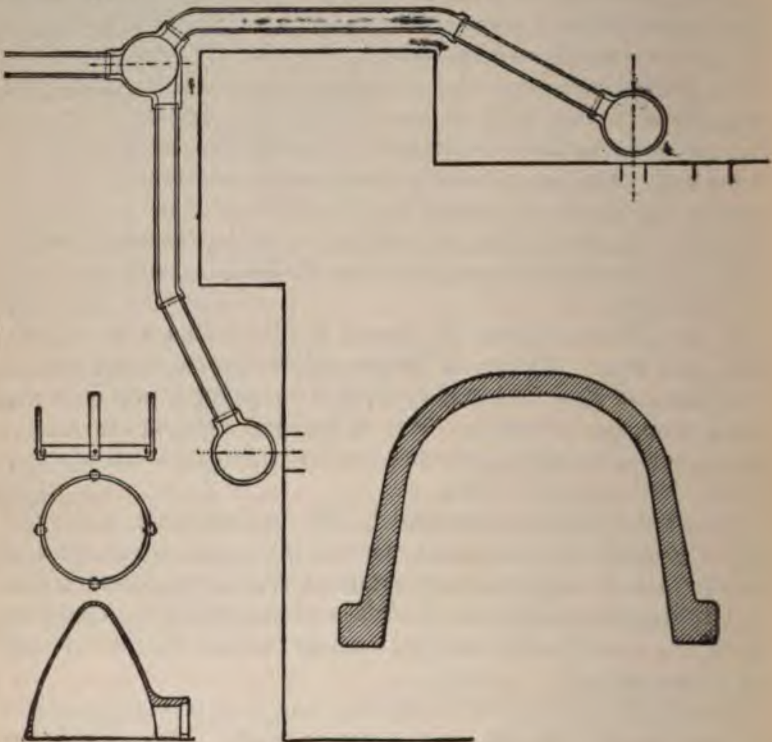


FIG. 67.

An inclination of one-half inch to the foot is found sufficient for well-smelted reverberatory slag, as it comes in such a volume.

The skimming of the slag is the weakest point in modern reverberatory practice. It wastes more time and heat than any other ordinary operation about the furnace. Even when executed simultaneously through skimming-doors at both front and side, it is a slow, unsatisfactory, and unmechanical operation.

The Boston & Montana Company, at its new smelter at Great Falls, has sought to remedy this difficulty by borrowing the tilting furnaces of the steel-makers. These are modified to suit the conditions, are fired with gas, and resting on a kind of rocking base, are easily tilted by a hydraulic piston. The slag is poured off with great speed and cleanliness, and, falling into a stream of water, is at once swept into the rapids of the Missouri River; while the matte, when it has accumulated sufficiently, is partially, or wholly, poured into molds. These tilting furnaces seem to be economical and sensible, and it is to be hoped that the company will soon allow a description of them to be published.

In the meantime, those who desire to study them more minutely have only to look into the steel practice, where many hints may be gained that will prove invaluable to copper metallurgists.

Yet another method of rapidly emptying the reverberatory hearth has occurred to various metallurgists. I know of no one who has given more serious attention to this subject than Mr. C. M. Allen of Butte.

He has designed a large, cast-iron box, divided into two unequally sized compartments, which are connected below something in the fashion of the siphon-tap of the Orford copper-furnaces. The difference in specific gravity of matte and slag will thus enable him to run off each of these substances from its separate compartment at different levels.

The invention is not yet completed, but seems to have merit. A still more radical departure would be the constant drawing off of matte and slag from a reverberatory furnace, and the constant feeding of a stream of ore; thus changing an intermittent into a continuous process. The vast body of molten material at so high a temperature would seem capable of receiving and smelting a considerable stream of fresh material, the access of heat necessary being constantly supplied from the ordinary fire in the grate.

The main difficulty would seem to be the impossibility of fettling the sides of the hearth without frequent emptyings of the same and consequent interruptions of continuity. The corrosion would be confined to a single line, just at the surface of the slag. Yet at this point, the inner lining of the furnace would soon be so deeply corroded, that it would probably have to be tapped dry in order to allow the proper amount of fettling to be done.

As the specific heat of slag is considerably higher than that of matte, it might be worth a trial to tap off all but a few tons of matte from underneath the slag-layer, and drop the fresh charge into a slag-bath instead of into a body of molten matte. Of course, a certain amount of slag would be withdrawn at each charge, so that the furnace should not become too full. Still, it is likely that the lesser specific gravity of the slag would offset the advantage of its greater specific heat, as the fresh charge would not be floated up so powerfully as it is upon a hearth-full of matte.

I have always preferred to run with a hearth over-full of molten material, and protect my two skimming-doors with dams of fire-clay. When the furnace is ready to skim, and after a thorough stirring and a quick heat to settle the matte grains, these dams are gradually rubbed down with a bar, and two-thirds of the slag flows off quietly by itself, without putting a rabble into the furnace at all.

The matte is now usually tapped into cast-iron, or in some cases steel molds. This does away with a great deal of dirt and confusion about the furnace, and relieves the men of a tedious and disagreeable task in making up the sand-beds again.

At Great Falls, the tilting-reverberatories pour their matte directly into a clay-lined ladle, in quantities of about five tons. This ladle is picked up by an overheard electric crane, running the entire length of the building, and is poured into one of the Bessemer converters, where it is blown up to 99 per cent. copper. This is poured directly in the shape of anode-plates, and goes to the electrolytic plant for the recovery of the silver and gold, and the precipitation of the copper as metal of the highest quality. It would be difficult to imagine a more economical arrangement than this, when it comes to handling matte on a large scale.

LABOR ON REVERBERATORIES.

The saving in labor by the use of large reverberatories is considerable. Yet the labor employed in running them cannot be well stated in general terms, as so much depends upon the manner in which the slag and matte are handled, how the coal and ore are delivered, and whether the latter is charged hot or cold; whether the charge is a basic one, and consequently cuts the furnace badly, as well as augmenting the amount of foul slag to be culled out and resmelted.

In a general way it may be said that a furnace smelting 50 tons, mostly of calcined ore, per 24 hours, will require per shift of 12 hours, one fireman, one-half the services of a skimmer, and one-half the labor of a man to handle foul slag, and assist generally. Aside from this, a small fraction of a man's time will be consumed in bringing tools to and from the smith's shop, and in similar little services. This is a trifle more than two men to the 12-hour shift, or, according to Butte wages, about $8 per 12 hours, making 31¼ cents per ton of ore smelted. This does not include bringing the ore or coal to the furnace.

The cost of labor per ton of ore is therefore reduced to about as low a point as is reached on well-managed blast-furnaces.

The character of the ore, and the comparative cost of raw and coked fuel must be the main items in determining which type of furnace to select in each individual case.

The blast-furnace usually makes a slightly poorer slag than the reverberatory. But this advantage may be outweighed by the fact that it also produces a somewhat poorer matte from the same charge.

Other things being equal, a siliceous mixture, or a charge consisting largely of finely-divided ore, would usually induce the selection of the reverberatory process.

DUST-CHAMBERS.

Where the smelting-mixture is rich, especially in the precious metals, a sufficient set of dust-chambers is an excellent investment, providing they are not allowed to interfere with the sharp draught so essential to quick smelting.

The method of forcing the escaping gases to pursue a zigzag course, over one partition and under the succeeding one, is very unwise. It is wrong in theory, and most unsatisfactory in practice. Instead of a placid body of stagnant air in which the dust-particles can remain completely quiescent after they have once fallen, we have in the zigzag chambers a fierce draught that sweeps the ore-particles along with it, and gives them little opportunity to accumulate in quiet.

A common, simple, and effective dust-chamber consists merely of a large, deep flue, with transverse walls built up from the floor part way to the roof. Between the upper edges of these cross-walls and the arch of the flue there is ample space for the quiet passage of the gases. The dust-particles drop, according to their weight, and as soon as they escape from the current of moving gases they fall quietly into the compartment below them. Here the air is absolutely dead, and they remain undisturbed until removed for further treatment. If warranted by the amount of flue-dust collected, the floor of the chambers may be made sufficiently sloping to discharge directly into a car alongside.

The charging of a reverberatory furnace by means of hoppers, especially when hot calcines are used, is accompanied with a dense cloud of dust, which, if collected, will usually be found to assay considerably higher in all the valuable metals than the average ore. The opening of the furnace-doors does not check the draught sufficiently to prevent a considerable portion of this fine dust being carried up the stack and lost; and from a few imperfect experiments that I have made in collecting this dust from a given surface, I am inclined to think that the loss in this way is often much greater than is suspected.

A simple device that I first saw, many years ago, in Mr. R. Pearce's furnaces, lessens this loss very considerably, and also checks the violent draught through the furnace, that cools the hearth so unnecessarily. It is simply a rectangular opening, say 16 by 30 inches, in the stack, on the side opposite the flue-entrance, and a short distance above the latter. A light, counterweighted iron door slides up and down in the iron frame of this opening. When the slide is raised, the air streams with great velocity into the hot stack, almost completely cutting off the draught in the hearth, and thus giving the finer ore-particles an opportunity to settle completely.

CONSTRUCTION OF REVERBERATORY SMELTING-FURNACES.

The excavation for the foundations should be 18 inches in every direction larger than the proposed furnace, allowance being made for the space occupied by the stack or down-take at one of the front corners. A depth of 4 feet from the floor level is sufficient, and a permanent drain should keep the pit free of water. Exceptional circumstances may require a greater depth of so much of the excavation as corresponds to the foundation of the stack. Two longitudinal walls are now laid in such a manner that a 5-foot space is left under the main body of the furnace, extending from the back of the ash-pit to a point directly under the future front wall of the furnace.

This is arched over with two 4-inch courses of red brick, upon which come one or two 4½-inch courses of fire-brick. The bridge wall and two lateral walls of the ash-pit are also begun from the same level. It is also well to carry up the side and front walls of the furnace from the very bottom, using red brick for all underground work, and filling the space between and outside of the walls with stone or slag, broken *in situ* with spalling-hammers, and firmly united with liquid mortar, or by pouring in the pots of slag as they come from the blast-furnace.

It is quite customary to fasten the looped tie-rods for the perpendicular buckstaves by merely bending a hook at the end of the rod and building it into the wall, trusting to the weight of the superincumbent mason-work to prevent their drawing out. It is a much safer plan to introduce the tie-rods at a lower level, giving them sufficient length and inclination to pierce one of the central longitudinal walls, and providing each with an eye through which passes a long, continuous bar of iron, which thus firmly holds all the tie-rods belonging to one side of the main body of the furnace. This bar is fastened to its fellow of the other side by a few short cross-rods, and the lower set of loops is thus firmly held in place, and far below any chance of being melted in two—an accident that would certainly occur in the course of time if they crossed the entire furnace *above* the subterranean arch.

The enclosing walls of the furnace having been built to within a foot of the floor surface, the hearth proper of the furnace is laid in the shape of an inverted arch, its lowest point in the center being in contact with the upper convex surface of the 4-foot subterranean arch, while its sides rise at the rate of about half an inch to the foot. It is also slightly arched longitudinally, and should be well keyed and grouted, as it is intended to be so constructed as to prevent the possibility of its being floated up by any breaking through of the molten contents.

The hearth is now enclosed by side-walls of fire-brick, 9 inches thick, which support the arch when the proper height is reached. These are encased by strengthening walls of red brick, while they are protected on the inside by a 9-inch lining of fire-brick, which can thus be renewed, when necessary, without interfering with the arch.

Two heavy, vertical cast-iron plates support the hearth at either end—the "conker plate" giving strength to the bridge-wall, while the front plate is placed just below the front door, the narrow horizontal skimming-plate resting upon it and determining the eventual thickness of the sand-bottom.

The bridge-wall is a massive structure of fire-brick, perforated by an air-passage about 3 inches wide, which has the conker plate for its anterior wall, while a lighter casting forms its posterior boundary. A large blow-hole or opening for the admission of air may be left on each side of the furnace in the angle formed by the posterior wall of the main portion of the structure and the wall of the fire-box. These orifices are used only when an oxidizing atmosphere is desired, as in the concentration of matte, the making of blister copper, etc., and can be tightly closed with clay under ordinary circumstances.

The fire-box is enclosed with a 9-inch wall of fire-brick, which may be strengthened by a casing of common brick, if desired.

Where coal is used for fuel, particular attention should be given to placing the grating-holes (that is the orifices in the sides of the fire-box just above the grate, through which bars are introduced to cut away the clinkers) in a convenient position.

The arch is best constructed of "Dinas" or silica brick, which last much longer than ordinary fire-brick, and should have a rise of an inch to the foot, pitching downward quite abruptly from a point slightly anterior to the bridge-wall, until it approaches to within 12 or 14 inches of the skimming-plate at the front door. Its shape, as well as the size and proportions of the space between bridge and roof, has much to do with the heating qualities of the furnace, and must vary with the character of the fuel and with other local conditions. The extreme front row of arch bricks, forming the posterior wall of the flue opening in the roof, is called the "vulcatory," and from its situation is so exposed to wear and heat as to require frequent renewal.

The flue opening itself is of a trapezoidal form, being enclosed laterally between the two converging walls of the hearth, while it has the vulcatory for its posterior and the front wall of the furnace for its anterior boundary.

Its size and proportions are matters of paramount importance, as the heating capacity of the furnace, as well as its consumption of fuel, depends largely upon it and upon the size and shape of the flue proper, that is. the canal connecting the hearth with the chimney.

No precise rules can be laid down in this matter for the guidance of the inexperienced, as each individual case must be judged upon its own merits until constant experimenting has determined the question.

The uncertainty and difficulty pertaining to this matter may be best appreciated when it is known that, of half a dozen furnaces in the same building, constructed from the same plan and apparently identical in every particular, fed with the same fuel, and smelting the same ore, no two behave in the same manner, and therefore each must have the size and shape of its flue suited to its needs. In general terms, it may be stated that a large flue will cause a greater consumption of fuel and a quicker heat, unless a certain limit is overstepped, beyond which the fuel will be burned without a corresponding rise of temperature. It is quite obvious, therefore, that the economical smelter will seek to throttle his flue to the greatest possible extent compatible with the rapid production of the required temperature. The flue should be narrowest at its junction with the furnace, and expand considerably as it enters the stack, having at least 50 per cent. greater area at the latter point than at the former. Its size is altered by introducing or removing a little dam of sand at the end nearest the furnace, one of the slabs with which it is covered being removed for that purpose. When experiments of this nature are executed to determine the most advantageous flue area, it is important that the change in size should be sufficient to produce some plainly marked effect, either for the better or the worse; otherwise, it is a mere groping in the dark. The weather, force or direction of the wind, and general condition of the atmosphere, may often produce an impression sufficiently powerful to entirely mask the changes brought about by the alteration of the flue area, so that a considerable period may be necessary to properly estimate the good or evil resulting from the efforts of the smelter.

A row of small openings should be left in the arch over the anterior edge of the fire-bridge; an arrangement that ensures the combustion of the gases in the hearth where they are needed, instead of in the chimney, where they are destructive.

Perhaps the most important portion of the furnace is its ironing, for its life largely depends upon the skill and intelligence displayed in holding it together. This subject is so familiar to all students of metallurgy, and the arrangements of all the iron work is shown so plainly in the drawings, that it may seem superfluous to dilate upon it. But a few practical hints may be of service.

Railroad iron, unless used in excessive quantity, is not strong enough to be used as buckstaves for the large, modern reverberatories. I-beams are far preferable, their shape giving them a very much greater stiffness per pound of weight. In the furnace illustrated, 8-inch steel I-beams are used, connected by 1½-inch tie-rods, which are all provided with turnbuckles, the diameter of the rod where the thread is situated being 1¾ inches, and the thread being cut very deep and, if practicable, with a considerably flatter pitch than the ordinary standard.

Instead of continuing the lower, lateral tie-rods through to the vault, they terminate in an anchor-plate just inside the inside walls, and they are reinforced by a heavy flagstone or slag block built against the buckstays at the level of the floor. This also greatly shortens the span of the beam.

A light I-beam is embedded in the brick-work opposite the skewbacks of the arches, and their lateral thrust is thus equalized among the buckstays. The longitudinal thrust of the hearth is received, respectively, by the heavily-ribbed conker plate and by the front plate, which convey it to the buckstays at either end. The chimney is entirely independent of the furnace.

In every kind of furnace construction, the plates, buckstays, etc., should be set in position and the brick-work built up snugly against them. This is the only way in which the mason-work can obtain a firm and proper bearing against the supporting iron-work. All the tie-rods and plates, and especially those portions of them that are buried in the mason-work, should receive a thick coating of hot coal-tar.

In addition to the red brick of good quality that may be used for the exterior casing of the furnace, three kinds of fire-brick are properly required to build a reverberatory smelter. These are:

1. Silica brick (Dinas, Mt. Savage, or similar brands), for the main arch, flue, and the greater portion of the side linings of the hearth.

2. A more basic (aluminous) and stronger brick, such as the Stourbridge, for the portions of the furnace that are exposed to mechanical wear or chemical corrosion from the fluxing action of the charge or of the ash of the fuel. Such portions are the lining of fire-box, and the lower parts of side-lining and front wall; also, usually, the stack lining.

3. A cheaper fire-brick, of good strength, for the arches over the vault, the inverted arch under the hearth, the walls of the furnace below ground, and all such portions of furnace and stack as are not exposed to the direct flame, and yet can scarcely be constructed of red brick.

All fire-brick should be laid with a thin mortar consisting of finely ground, one-half raw and one-half burned (or ground fire-brick). The mixture should all pass a 30-mesh screen, and the joints should be as tight as possible, each brick being tapped down to its bearing with the hammer. There need be scarcely any cutting of brick during the work. This costly and unsatisfactory operation can be mostly avoided by procuring a few proper "shapes" of each variety of brick. Only ordinary stock shapes are required, such as bullheads, wedge-brick, jamb-brick, side and end skew-backs, soaps, and splits.

The crown, or main arch, of the furnace will require a rise of between three-eighths and five-eighths of an inch to the foot. The brick in this arch are laid perpendicularly, or nearly so—the 4½-inch side being parallel with the longitudinal axis of the furnace. Every alternate course is begun with a "split" brick, which, being only 1¼ inches thick, causes every row across the furnace to break joints.

The silica brick for the crown are carefully assorted as regards thickness and hardness, and are simply dipped in a slurry composed of 60 per cent. fine fire-sand and 40 per cent. raw, ground fire-clay. Each brick is carefully tapped to a bearing with the hammer, a piece of board being interposed to protect the fragile brick.

The tie-rods are drawn up to a moderate tension before beginning the arch at all, and the row of key-brick along the center of the crown is driven in with sufficient force to completely relieve the wooden arch-pattern of all superincumbent load, so that it can be removed without force.

The brick that compose the arch, the bridge, the side-linings, and the fire-box lining should all be laid as headers, *i.e.*, 9 inches thick.

CHIMNEYS.

Under ordinary circumstances, it is probable that an independent chimney to each furnace is best suited to the somewhat fluctuating conditions of American metallurgy. Before enough advantage can be gained from the costly system of flues inseparable from a single, central stack to create a sinking fund to redeem them, it is highly probable that radical improvements in the process will have rendered them unsuitable for the new conditions; or else, that the construction of new lines of railways will have made it advantageous to remove the works to some different locality, or to suppress them altogether.

Where a stack is built for each furnace, the copper reverberatory chimney is a small and comparatively inexpensive affair; and, after elaborately calculating the speed, volume, and temperature of the gases, the skin friction of the walls, etc., the experienced metallurgist lays his papers one side and builds his stack 65 feet high and of a little greater area than accumulated experience has shown to be necessary. Assuming the area to be sufficient, and there being no surrounding eminences to cause descending air-currents that may check the draught, experience has shown that the difference in weight between a column of gases 60 to 65 feet high at a temperature of 700 degrees Fahr. (371 degrees C.), and an equal column of the outside air,is sufficient to create as rapid a draught as can burn coal to advantage under conditions prevailing in the ordinary reverberatory of to-day.

With a central stack common to several furnaces, the gases are greatly cooled during their passage through the long flues, and a height of 120 feet, or more, must be obtained. In order to lessen friction and to allow for the accumulation of dust, the connecting flues should have an area far beyond the apparent requirements of the draught, and care should be taken to have the different flues join each other at as small an angle as possible, else some one or two furnaces will "steal the draught," and the others will suffer accordingly. Where the main flues enter the stack by different openings, the shaft should be correspondingly divided by vertical partitions to a height of several feet above the flues.

High chimneys should be round, and built from the inside. Lower chimneys, up to 80 feet, should be square, and built from the outside, using an ordinary scaffolding of poles or scantling, braced diagonally with boards. The red brick casing on the outside may be laid in lime mortar; the rest of the stack should be laid with clay mortar consisting of raw, ground fire-clay, with a sufficient addition of burned fire-clay (or ground fire-brick), to make it leave the trowel easily.

The stack must have a solid foundation, and should be constructed so as to leave a free space between it and the furnace, in which the buckstays of the latter may be placed. The stack belonging to the reverberatory shown in the accompanying illustrations (Figs. 60 to 63), has a casing of hard red brick 8 inches thick, an inner, or main wall of fire-brick 9 inches thick, an air space 2 inches wide, and an inner lining of fire-brick 4½ inches thick. At a height of 40 feet, the outside casing and inner lining are suppressed, the main 9-inch wall of fire-brick rising alone for the next 20 feet, and the top section of 5 feet, consisting of only 4½ inches of fire-brick. There is thus an 8-inch offset on the outside at 40 feet, and an 11-inch (9+2) and a 4½-inch offset on the inside. The stack tapers inside and outside at the rate of ⅛ inch to the foot, and thus loses one inch in size for each 4 feet in height, or about 16 inches in total. It gains $(2\times 11)+(2\times 4\frac{1}{2})=31$ inches in size by the internal offsets, so that it is considerably larger at the top than at the bottom; a circumstance that seems to be favorable to the draught in spite of the contraction of the gases due to cooling.

A rectangular opening 16 by 30 inches should be left in the stack opposite to the flue entrance. It is provided with a counter-weighted sheet-iron sliding door, and should always be raised when the furnace is not smelting. This effectually checks the draught, and thus prevents the undue cooling of the furnace, as well as loss in dust while charging.

There should be an ample free space about a reverberatory furnace, at least 15 feet on the tapping side, and the same distance in front, while a space of 12 feet on the charging-door side will suffice. This should be well drained and paved with brick on edge, or with cast-iron plates.

The arch being completed and the wooden pattern removed, the furnace is taken in charge by the smelter, who, with the blacksmith's aid, proceeds to the proper tightening of the tie-rods; the side buckstaves having been already sufficiently drawn up to keep the arch in place. This process has been described in the chapter on "Calcining Furnaces," and presents no peculiarities. The empty hearth should be covered with a 2-inch layer of fire-clay, to prevent adhesion of any metal that may possibly make its way through the sand-bottoms.

A small fire may be at once built on the surface of the clay stratum and in the ash-pit, and should be maintained for at least four days, slightly raising the temperature, until, at the expiration of this time, a dark-red heat is attained, and the cessation of aqueous vapors from the side walls and subterranean arch shows that every particle of moisture is removed.

The grate-bars are now placed in position, being mere rods of inch square wrought-iron—and the fire, being shifted to its proper position, is gradually urged for 12 hours or more, until the whole interior is of a light-red heat.

Then, and not until then, should the material for the smelting-hearth be introduced.

This consists essentially of silica, and may be in the shape of well-washed beach sand, or crushed sandstone, or of pulverized quartz, first roasted in lumps and quenched while hot, to impart a high degree of brittleness and greatly facilitate its crushing.

A beach sand employed for this purpose in Swansea, and analyzed by Percy, had the following composition:

	Per Cent.		Per Cent.
Silica	87.87	Magnesia	0.21
Alumina	2.13	Carbonic acid and water	2.60
Sesquioxide of iron	2.72		
Lime	3.79	Total	99.32

This is not so refractory as the crushed sandstone employed by some of the eastern American smelting-works, or the pulverized quartz used for reverberatory bottoms in Butte, Montana, which, according to the author's tests, contain respectively, when dry, 95.3 per cent. and 97.2 per cent. of insoluble residue, presumably silica.

Two methods are pursued in making reverberatory hearths. Either the sand chosen contains enough bases to be slightly fusible, or a small proportion of crushed slag or other similar substance is added so that the sand may become slightly agglomerated by the intense heat to which it is subjected; or secondly, the material selected is practically infusible, and the cementation of its particles is effected by smelting small and repeated charges of fusible material upon the slightly hardened surface of the same, until it is solidified into a hard and impermeable mass.

The author prefers a combination of these two systems, using the first method for the lower hearth, and the second for the upper or true hearth, as it is usual to put in two separate hearths, the upper one being comparatively thin, so that it can be easily removed when worn out.

The total height from the floor of the furnace to the upper surface of the skim-plate being perhaps 40 inches, the lower hearth (including the clay bottom) should have a thickness of 14 inches and the upper of 26 inches, both of them being somewhat concave in shape, so that a basin is formed some 14 inches deeper than the skim-plate in the center, and sloping from every direction toward the tap-hole.

The size of the hearth material is a matter of less importance than is often supposed, provided that, if at all coarse, sufficient fine dust is present to fill all interstices and prevent porosity.

If good crushing facilities can be had, it is well to pass everything through a 10-mesh screen, but the author has used even a 5-mesh without evil results.

This, of course, refers to sandstone or quartz rock. Natural sand usually requires no sizing process, unless mixed with gravel. The utmost care should be taken to prevent the introduction of any foreign material, especially of an organic nature, as the gases generated therefrom may easily cause a ruinous flaw or blister in an otherwise perfect hearth.

Such unfortunate results, however, are usually counteracted by the thorough calcination that all sand must undergo previous to the final melting.

A sufficient amount of the sand—usually from 6 to 10 tons—being thrown into the heated furnace (either as such, or mixed with from 3 to 5 per cent. of pulverized slag), a moderate fire is maintained, while a steady stirring and rabbling is kept up through the side and front doors until every particle of moisture and carbonic acid and other gases is expelled, and the heat gradually raised to such a temperature as to insure the decomposition of all organic material. This operation may require from 3 to 8 hours, according to the nature of the material. Both the temperature and time are matters of great importance, having a marked effect upon the final result; but can only be learned by experience, as they vary with each different sand.

Toward the close of this period, the sand is gradually brought into the proper shape for the bottom, and thoroughly pressed and stamped into place by means of long paddles and stampers, worked through the door openings. No great pains need be expended upon the lower hearth as it will, of course, be entirely covered and its shape obliterated by the superior layer.

The doors are then closed, the tap-hole bricked up and covered with a heap of sand, and every crack and orifice about the whole furnace completely stopped and closed. The fire is gradually urged until the highest possible temperature is reached and maintained for a couple of hours, the entire period of heating requiring from 6 to 14 hours, according to the heating capacity of furnace and fuel. The interior condition of affairs is watched through a peep-hole in the front door, which is provided with a clay plug. After a proper maintenance of the highest temperature, the fire is gradually slackened and the furnace cooled down. This operation demands the greatest care and circumspection, as the premature opening of a door, or a sudden draught of cold air, may cause the appearance of a crack or blister in the porcelain-like surface of the sand-hearth. Several hours must elapse before the doors can be taken down and the results of the operation inspected. The interior of a reverberatory furnace under these circumstances is quite an interesting sight. Long stalactites of molten fire-brick hang down from the arch over its entire surface. The side walls are not only glazed, but actually fused until they begin to soften to a considerable depth, and the hard and glistening semi-fused surface of the new hearth is strewn with fragments of brick from the crown, and little heaps of molten fire-clay corresponding to the pendent stalactites.

Unless very serious cracks exist, no notice need be taken of them, and blisters and irregularities may be entirely overlooked, as the upper hearth is to bear the brunt of the work. After slow and perfectly even cooling to a dark-red heat, about 3,000 pounds of moderately basic, fusible slag, crushed to the size of chestnuts, are spread over the entire surface, being charged by means of long-handled paddles, and on no account thrown in carelessly, to be subsequently leveled with rabbles, as is often done. The doors being again tightly closed, the slag-charge is quickly smelted down, two hours being amply sufficient for this purpose. This layer of slag will be entirely absorbed by the porous sand-bottom, which, after a second cautious cooling, should be again charged with a somewhat larger burden of slag, with which are mixed a few hundred pounds of low-grade matte (30 per cent.). After this is melted down, a considerable portion will probably be found in a pool near the tap-hole, from which it should be immediately evacuated. If the furnace is to be used for concentration work, or especially for the production of blister copper, still another charge should be melted on the lower hearth, consisting principally of matte of the same grade as the former, and should be tapped as soon as sufficiently liquid. In this way, the lower hearth will be pretty thoroughly saturated with matte of low tenor, thus preventing the absorption of an equivalent quantity of richer metal in case the same should penetrate from the upper hearth. The tying up of a large amount of copper may thus be guarded against.

After a final cooling, the sand to form the upper hearth is thrown in, and, after a careful calcining and leveling, should be stamped into place with the utmost care, sloping gently toward the tap-hole from every point.

The fusion is executed in the manner already described, and in addition, a careful watch should be maintained at the peep-hole to observe and remove any fragments of brick that may fall from the arch during the early period of the fusion. After the softening of the sand has once begun, no further manipulation is permissible.

The cooling after the fusion must be executed with extreme care, to prevent cracking and blistering, and as soon as a dull-red heat is reached, the slag and matte charges already enumerated should be successively melted, with alternate periods of cooling.

As a final preparation before introducing the first ore-charge, a small quantity of finely crushed slag should be thrown around the entire edge of the hearth at the junction of sand and fire-brick.

Above this, a thick bolster of "fettling" (mixed fire-clay and crushed quartz) should be tightly forced into the angle between hearth and side-walls, including the bridge-wall.

An hour's brisk heat will dry and consolidate the fettling, and the regular work of the furnace may begin—small charges being used at first, and no large quantity of metal allowed to accumulate before tapping.

MANAGEMENT OF FURNACE.

The experienced furnace-man constantly watches his furnace with reference to the safety and condition of its bottom. After a few hours' firing on a fresh charge, the workman introduces his rabble, and by the feeling of the sand when gliding over the bottom, determines at once the condition of things.

If slippery and sticky, it indicates that portions of the charge still adhere to the hearth. These are removed as far as possible by the rabble, and dissolved by a short additional heat, until the rabble glides smoothly over a plane, granular bottom, which is the upper surface of the hearth proper. After this condition is once attained, every additional moment of high temperature is not only wasted, but is positively detrimental to the hearth, which lacks the protection of the semi-fused ore, the liquid matte soon attaining a high temperature, and, if exposed to the air, boiling in a manner that may prove highly dangerous to the bottom. This is the case when concentrating matte or making blister copper—operations very severe on the bottom, but rendered less dangerous by being conducted at a lower temperature than is required for smelting proper.

Equally detrimental may be a high temperature with a small charge, where the unprotected portions of the bottom may become so softened as to rise in large flakes, being literally floated up by the superincumbent metal.

Any large piece of iron, such as a rabble-head, may cause a hole in the bottom, and, in endeavoring to float up an old bottom, nothing is more effective than the introduction of a number of large fragments of old iron. A bottom may be often patched to advantage when only locally damaged. When any such condition is discovered, the hearth should be immediately emptied, and the damaged portion, which usually shows a decided cavity or depression, should be most carefully emptied, fresh sand being repeatedly introduced and again removed with the rabble until it is completely dry. The hole should then be filled and leveled up with ordinary bottom sand, which must be fused and saturated with the same precautions as in the case of the original bottom. In this way, a bottom may often be saved for many months at a very slight expense.

In direct connection with the management of the bottom is the proper fettling of the furnace. The entire life of the side walls and safety of the bottom depend upon the care and conscientiousness observed in maintaining the dam that incloses the molten, liquid pool and protects the fire-brick. In default of this safeguard, the side walls are quickly undermined, a groove several inches in depth being cut into the mason-work during the smelting of a single basic charge. Nothing then remains to prevent the descent of the metal between the wall and bottom until the latter is floated up and ruined, and a large amount of copper temporarily lost. The best fettling is formed of pure, white quartz, crushed through a 3-mesh screen and mixed with sufficient plastic fire-clay to form balls, which may be placed at exactly the required point, and forcibly pressed and molded into place. The quartz may be replaced by ordinary bottom sand, which, however, is less permanent and solid. When smelting basic ore, the hearth may require fettling after every few charges; but with a quartzose mixture, days may elapse without any necessity for renewal. Safety in this particular is only obtained at the expense of constant watchfulness.

The size of the fire-box and depth of grate below the upper surface of the bridge are very variable factors, depending upon the quality of the fuel and degree of temperature.

The best and most economical results are obtained by the use of the clinker grate, which is virtually a gas generator, a deep layer of clinkers being maintained upon the grate-bars, penetrated by numerous openings through which the air passes, being heated to a high temperature before it unites with the gas generated from the coal, which lies upon the upper surface of the bed. A certain proportion—from one-third to one-half—of caking coal is required for this method of combustion, and the grate-bars must be at a much greater depth than for ordinary non-caking fuel.

Lignites, or any free-burning, non-caking coal, require a shallow grate and a large flue, while wood behaves in much the same manner, requiring, however, the introduction of air through holes in the roof above the bridge, on account of the great volume of combustible gases generated.

It is almost impossible to give any general rule for the amount of fuel required for a reverberatory furnace. When engaged in smelting ores, a much larger quantity must be consumed than in making blister copper or in the refining process, where only a very moderate temperature is needed for a considerable portion of the time.

It varies at well-managed works in the United States from 2.5 to 5 tons ore smelted per ton of coal.

The table on page 489 gives a good idea of the amount of fuel consumed in smelting somewhat difficult ores.

The main features that distinguish modern reverberatory practice, and the means employed for rapidly handling the vast quantities of slag produced in furnaces smelting from 35 to 70 tons of ore per 24 hours, have already been alluded to in the earlier pages of this chapter. The accompanying illustrations show one of the large Argo furnaces with its slag-launders and settling-pots. In the cut, the furnace is skimmed from only one side door in addition to the front door; but where the slag is very large in quantity (over 10 tons per charge being skimmed at Argo), much time is saved by skimming from two doors on each side simultaneously, the pots and gutters being arranged accordingly.

Such a bulk of slag as is now made at reverberatories cannot be removed conveniently or economically by simply skimming it into sand-beds directly at the furnace and removing the pigs in barrows after they are sufficiently chilled to stand handling. There is neither space for the pigs nor time for them to cool, and access to the furnace would also be difficult. There are three methods available for the handling of this slag. These are:

1. Running it directly out of the building in iron gutters, one section of the gutter being hinged, so that it can be raised up where it crosses the passageway. The slag being once outside of the building can be disposed of, according to convenience, in various ways, such as:

(*a*) Running it into sand-molds, the pigs being allowed to cool, and then carried away by railroad, as at Argo.

(*b*) Running it at once into molds for the manufacture of slag-brick.

(*c*) Running it away with water, which requires a large and powerful stream of water to safely break up the somewhat chilled slag.

(*d*) Running it into large pots or pyramidal iron boxes, as at iron blast-furnaces, the vessels running on a track and being moved by mule or steam-power.

2. Granulating it at the furnace by a stream of water, as at the Parrot and the Great Falls Works in Montana. The volume of water that will flow through a 6-inch pipe under a 14-foot head is found sufficient to granulate the slag during the skimming of one reverberatory, the inclination of the sluice depending upon the specific gravity of the slag, but averaging half an inch to the foot. The granules may be treated as is described when treating of the removal of blast-furnace slag by water.

3. Running it into large pots at the furnace, the pots being pulled by mule or steam-power. This is likely to be a convenient method in many cases, and is, perhaps, the most economical to install. The large, double Nasmyth pots, mounted on a truck, may be employed, and, if arranged to tap from a hole near the bottom, may be used for making slag-shells, in which any stray particles of matte become concentrated, as at the silver-lead furnaces. The use of these pots is attended with the disadvantage of having to run them on a lower level than the furnace, which cuts up the floor of the building.

COSTS OF RUNNING A REVERBERATORY FURNACE.

The costs of smelting in reverberatory furnaces have been so lessened since the first publication of this work, and the whole system of manipulation and management has been so simplified, that my older estimates possess but little value for our present conditions. Nor do I feel it best to replace them in the same form with more modern figures, as aside from the great difference in prices at different localities, the expenses of running a reverberatory vary so extraordinarily, according to quality of coal, local custom as to what constitutes a day's work for a smelter, convenience of plant for handling ore and products, etc., that it seems to me hardly worth while to attempt to reduce the costs of these items to any exact figures. It will be of more general utility to mention the average amount of labor, fuel, etc., that are now generally required to run a reverberatory under American conditions. I incidentally add the prices that prevail at Butte, Montana, which is our great reverberatory smelting center for copper ores.

Omitting furnaces of great capacity—50 to 70 tons ore per 24 hours—the average amount of labor employed at an ordinary reverberatory, smelting say 30 tons of cold ore per 24 hours, and having the slag removed by a steam of water, will be, per shift of 12 hours:

One fireman	$3.50
One-half skimmer	2.00
One-half laborer	1.50
Total labor	$7.00

This is at the rate of about $0.47 a ton for labor.

Such a furnace will burn about 8 tons fair coal per 24 nours, at $6 per ton, making $1.60 per ton of ore for fuel. This is of course the one great expense of reverberatory smelting in districts where coal is comparatively costly, as at Butte.

Blacksmithing and other slight expenses about the furnace, averaged from various works, amount to $0.17 per ton of ore.

Repairs, including bottoms, vary greatly under varying conditions, but may be averaged at $0.16 per ton of ore.

To this must be added $1,000 per year, or say $0.10 per ton of ore, as a sinking fund to replace the furnace in 5 or 6 years.

The totals, excluding general expenses and interest on investment, are, per ton of ore smelted:

Labor	$0.47
Fuel	1.60
Miscellaneous	0.17
Repairs	0.16
Sinking fund	0.10
Total	$2.50

The transportation of the cre to the furnace will cost from \$0 08 to \$0.15 per ton, and the items of superintendence, foremen, assaying, office expenses, insurance, pumping water, illumination, resmelting slag, etc., will add an amount to the above total varying from \$0.30 to \$1 or more per ton of ore. It is in these latter items that large, well-arranged works with ample capital can greatly excel smaller plants that are cramped in their means.

COST OF BUILDING A MODERN REVERBERATORY SMELTING FURNACE.

The cost of a modern reverberatory, based on a compromise between Eastern and Western prices, and with a 14 foot by 24-foot hearth, substantially as shown in Figs. 60 to 63, will be about as follows:

COST OF REVERBERATORY FURNACE AND STACK.

Masons' materials—		
Fire-brick, 55,000 at \$35		\$1,925.00
Silica brick, 10,000 at \$50		500.00
Red brick, 40,000 at \$8		320.00
Raw fire-clay, 10 tons at \$8		80.00
Burnt fire-clay, 10 tons at \$9		90.00
Lime, 75 barrels at \$1.20		90.00
Cement, 50 barrels at \$1.50		75.00
Sand, 25 loads at \$1		25.00
Pattern lumber, 3,000 feet at \$18		54.00
Total masons' materials		\$3,159.00
Iron castings—	Pounds.	
1 front-plate	1,600	
1 main bridge plate	2,100	
1 secondary bridge-plate	660	
1 supporter for fire-box wall	800	
2 skimming blocks	176	
8 side-door plates	1,680	
3 enclosing fire-box plates	3,864	
Gutters and pots	3,960	
Miscellaneous	640	
Total castings at 2¼ cents	10.380	\$374.50
Steel I-beams—	Pounds.	
Buckstays, 480 feet of 7-inch I-beams at 20 pounds per foot	9,600	
" 30 " 8 " " 22 " "	660	
Skewbacks, 66 " 6 " " 17 " "	1,122	
Bearing and grating I-beams at 9.33 pounds per foot	224	
Total steel beams at 3 cents per pound	11,606	\$348.18

Wrought iron—	Pounds.	
Tie rods, etc., 500 feet, 1¼ inch round iron at 6 pounds per foot	3,000	
16 turnbuckles at 14 pounds each	224	
2 skim-door frames and fixtures	160	
20 grate bars at 30 pounds	600	
3 grate-bar rests at 70 pounds	210	
Miscellaneous	260	
Iron for stack—8 uprights, ⅜-inch x 1-inch x 65 feet at 2 pounds per foot	1,300	
112 cross straps, 2 inches x ¼-inch x 9 feet at 15 pounds each	1,680	
Total wrought iron at 2½ cents per pound	7,434	$185.85

Labor—	
200 days masons' labor at $4	$800.00
210 days helpers' labor at $2	420.00
11 days carpenters' labors at $3	33.00
12 days smith and helpers' labor at $5	60.00
45 days common labor at $2	90.00
Total labor	$1,403.00

Miscellaneous—	
Excavation (2,000 cubic feet and stone work)	$325.00
Grading and clearing	60.00
Incidentals and superintendence	200.00
Hoppers, complete	325.00
Total miscellaneous	$910.00

Résumé of totals—	
Masons' materials	$3.159.00
Iron work	908 53
Labor	1,403.00
Miscellaneous	910.00
Grand total	$6.380.53

SMELTING FOR WHITE METAL.

As the production of the higher grades of matte, of which *white metal* (from 70 to 75 per cent.) may be regarded as the type, by means of the fusion of calcined coarse metal with quartzose ores, presents no sufficient differences from ore smelting to demand especial notice in this very brief treatment of the subject, the older process of concentrating metal without the intervening calcination need be alone considered under this head.

This process is termed "roasting" by the English smelter, and denotes the gradual fusion of the coarse metal in large pigs, on the hearth of a reverberatory furnace, with the abundant admission of air.

It is seldom practiced in the United States on account of its extreme slowness and consequent great consumption of fuel and labor, but it possesses the advantage of great simplicity of plant, dispensing as it does with the entire crushing and calcining paraphernalia.

Despite the simplicity of the process, much experience is required to obtain the best results, as the exact degree of temperature at the differing stages of richness of the product has much to do with the rapidity of the concentration.

Experience has taught that the rapidity of this concentration stands in exact proportion to the richness of the matte operated upon. The explanation of this is, that the sulphur, which is almost the sole foreign constituent of the richest matte, is very easy of oxidation, while the iron, which increases with the decrease of copper, oxidizes with much greater difficulty.

The writer has given much attention to this subject in connection with futile efforts to effect what M. Manhès has now accomplished with his bessemerizing process. The following table gives, in per cent. of product, the result of his experiments, which extend over several years, many of them having been conducted for the Orford Copper and Sulphur Company, while the author was in its employ. They were made merely to determine the rapidity with which the grade of the matte increases by the ordinary method, and without any attempt at bessemerizing.

Great care was taken in all instances to insure the correct sampling and assaying of the substances under consideration.

It will be understood that the matte was charged in the shape of large pigs; melted down during the time indicated (in most instances, about five hours), and retained in a molten condition (in both stages with the free admission of air) for varying periods, samples being taken from time to time—after thorough stirring—to determine the progress of the concentration.

TABLE OF MATTE CONCENTRATION BY OXIDIZING FUSION—PERCENTAGES OF COPPER IN FRACTIONS OMITTED.

Matte charged.	When melted fully 5 hours.	6 hours.	7 hours.	8 hours.	10 hours.	12 hours.	14 hours.	16 hours.	18 hours.	20 hours.	22 hours.	24 hours.	26 hours.	28 hours.	30 hours.	32 hours.	34 hours.	36 hours.	48 hours.
16	16	17	16	19	20	20	21	22	21	...	...	23	...	...	23	...	...	25	29
21	23	22	...	...	...	25	...	...	27	...	...	27	...	...	...	...	...	...	...
33	37	41	...	39	...	41	...	...	41	...	...	44	...	...	49	...	...	...	...
41	45	...	...	47	...	53	...	...	54	...	...	58	...	...	...	...	...	...	...
50	55	...	...	57	...	59	...	...	61	...	...	61	...	...	64	...	...	...	...
58	62	62	62	61	61	62	...	65	...	65	...	67	68	...	...	...	...	...	...
63	67	...	...	70	...	72	...	...	75	...	...	78	...	...	84	...	...	...	...
69	73	73	74	74	77	78	77	82	85	...	...	89	...	...	94	...	98	...	...
74	82	...	...	84	...	88	...	...	94	...	...	...	99	...	...	...	...	...	...
80	86	...	...	89	...	93	...	98	...	...	...	...	...	...	...	...	...	...	...
86	94	...	...	...	...	...	99	...	...	...	...	...	...	...	...	...	...	...	...
92	96	96	98	99	...	...	...	99	...	...	...	...	...	...	...	...	...	...	...
96	98	...	99	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...

THE MAKING OF BLISTER COPPER.*

This very beautiful and interesting operation is entirely of English origin.

The furnace used for this purpose, while an ordinary reverberatory as regards size and shape, should be very strongly ironed, to withstand the large charges used in our modern practice, while its bottom should be smelted in with peculiar care, and its upper layer should be thoroughly saturated, before used, with metal of the same grade as blister copper (from 96 to 98 per cent.), to prevent the certain annoyance from the rising of bits of the poorer matte just at the completion of the process, and the consequent adulteration of the whole charge of blister, which will require still further oxidation to remove the impurities. The lower bottom may be slightly saturated with lean matte to save expense.

The metal is charged in large pigs, the total weight depending principally upon its grade; for as a full bed of blister copper (from 8 to 20 tons) is usually desired as most economical, it is evident that a much greater weight of blue metal (62 per cent.) will be required than of white metal (75 per cent.); while pimple metal

* This method is now largely displaced in the United States by the process of bessemerizing the matte in converters, starting with a matte containing 50 per cent., or more, copper, and blowing it up direct to good blister.

(83 per cent.) and regule (88 per cent.) will lose still less in the process.

The technical names just enumerated apply to various grades of matte, each of which has its invariable characteristics, which distinguish it with certainty. The percentages given therewith are not absolute, but are subject to considerable variation, the writer giving such average figures as his own experience has determined for him.

As both economy and a due regard for the furnace bottom prevent the blister charges from covering too long periods of time, it is necessary to shorten the same by using either a less weight of matte, or insisting upon a higher grade at the outset.

The latter is the proper choice, as a small charge is almost certain to injure the furnace bottom by leaving a portion of it exposed to the direct heat of the flame.

The most advantageous lengths for the working off of a blister charge must depend largely upon local circumstances. From 24 to 36 hours will finish a full charge of rich pimple metal.

A similar weight of white metal may require 50 hours, which is quite long enough for the safety of the furnace, though a much greater length of time often elapses without harm.

As will be readily seen, it is impossible to work off a charge of blue metal within the prescribed limit of time, while even white metal extends the period most unpleasantly. It is therefore much better with metal of low quality to divide the operation into two stages—producing, for example, pimple metal or regule the first time, and bringing it up to blister copper by a second step. In this way full charges can be used without endangering the furnace, and many advantages are gained.

It is not well, however, to alternate the operations in the manner just suggested; but rather to keep the furnace on one grade of metal until a large amount is collected, and then take up the blister process and maintain it until all the concentrated metal is disposed of. In this way, the evil of attempting to make blister copper on a bottom saturated with poorer matte is avoided; and the exact amount of concentrated metal required for a full charge of blister can always be had.

The matter may be pushed a step further, by using a separate furnace for each operation, and positively interdicting the use of the blister furnace for any other purpose.

The operation of making blister copper is frequently executed by constantly maintaining the charge in a molten condition after it is once melted, and never allowing it to chill or "set," as it is technically termed.

By pursuing the latter plan, however, the danger to the hearth in long campaigns is greatly lessened, as it thus has a slight opportunity to cool, while the process is certainly advanced in a remarkable degree by the alternate fusions and chillings.

The belief that the operation of tapping causes a great gain in the grade of the matte, expressed among the Welsh smelters by the vulgar saying that "two tappings is worth one blowing," is contradicted by the following experiment, executed by the writer for the purpose of determining the truth or falsity of the common belief:

	Just before Tapping. Per Cent.		Just after Tapping. Per Cent.
No. 1 Copper	53.6	No. 1 Copper	53.2
No. 2 "	66.8	No. 2 "	68.0
No. 3 "	72.7	No. 3 "	70.2
No. 4 "	75.4	No. 4 "	75.2
No. 5 "	79.3	No. 5 "	79.1
No. 6 "	85.4	No. 6 "	86.2
No. 7 "	94.0	No. 7 "	93.2
No. 8 "	98.7	No. 8 "	98.5
Average	62.59	Average	62.36

A few test assays that showed remarkable variation one way or the other were discarded, and, as seen, the others actually show, on an average, lower, rather than higher, grade of the metal after tapping, which is doubtless merely an accidental circumstance.

The exact grade at which the blister copper should be tapped is a matter of much importance, as either too high or too low a copper presents physical qualities injurious to the requirements of the process.

Blister copper of just the right grade is highly "red-short;" that is, can be easily broken when red-hot. It is this quality that enables the furnace-man to break and separate his pigs of blister copper, which operation is rendered greatly more difficult by a very slight variation in percentage in either direction. The proper condition of the charge is easily made manifest to the experienced by ocular inspection, and the writer has endeavored to fix the limits that bound the grade of copper possessing this important quality.

Foreign impurities exert so much influence in this direction as to render impossible any exact establishment of such boundaries; but numerous tests have fixed the most favorable grade between 96¼ and 99 per cent.

An important precaution in the care of a blister furnace is the proper draining of the hearth and stopping of the tap-hole. Carelessness in this respect will permit a slight and unsuspected leakage during the entire period of blister-making, culminating in a mass of metallic copper filling the entire tap-hole, and, as has occurred to the writer, requiring the combined efforts of the furnace *personnel* and blacksmith employés for 12 hours to remove it.

The charge, being high blister, and just ready to tap, exerts a ruinous effect upon the furnace bottom during any such delay, and should be ladled out at once under similar circumstances.

The slag from the blister process, being very rich in oxide of copper, should be returned to some process where the product is of high grade, and not, as is often the case in this country, sent back to the ore smelting, where its copper contents are thrown back again to the condition of a base sulphide.

The following are examples of the composition of the slag and copper produced by this roasting-smelting for blister copper:

	Welsh "Roaster" Slag.*	"Roaster" Slags from Kaafiord.		Welsh Blistered Copper.	Blistered Copper from Kaafiord.†
Silica	47.5	36.0	Copper	....	99.2–99.4
Protoxide of iron	28.0	7.0	Iron	0.7–0.8	0.1– 0.2
Alumina	3.0	6.0	Nickel and cobalt	0.3–0.9	0.2– 0.3
Cuprous oxide, Cu_2O	16.9	43.2	Zinc	....	0.0– 0.02
Lime	....	2.7	Tin	0.0–0.7	... .
Magnesia	....	0.8	Arsenic	0.4–1.8	....
Nickel and cobalt oxides	0.9	4.9	Sulphur	0.1–6.9	0.1– 0.12
Oxide of tin	0.3	0.6			
Oxide of zinc	2.0	3.2			

* Le Play.

† Kerl, Grundriss der Metallhüttenkunde i., p. 215.

COPPER REFINING.

The only method of copper refining practised in the United States, or, in fact, in the civilized world, is the ordinary Swansea process.*

The purity of our local ores and the simplicity of our trade requirements have prevented the development of those marked variations that characterize the English process, where "best selected," "tough cake," "founder's metal," and many other distinct varieties are demanded and produced.

The great excellency of the Lake Superior copper, derived from pure metallic ores, has established a very high standard in our markets, and owing to the abundant supply of the same, manufacturers have not, until recently, found it necessary to study the behavior or familiarize themselves with the capabilities of other brands of copper for certain uses, but, feeling sure that if they bought the best they would be safe, have employed this unnecessarily superior metal for the manufacture of brass castings, and many other purposes where a poorer quality would have served equally well.

The most impure domestic coppers are often sufficiently argentiferous to repay a separate process by which the quality of the baser metal is improved, while the nobler is saved—of late, largely by electrolytic means.

The refining-furnace presents no peculiarities to distinguish it from the ordinary English reverberatory, except that it should be more strongly constructed; being provided with a massive front plate—below the skimming-door—as well as strong horizontal, lateral braces to strengthen the hearth, which, in addition to the enormous expansive force caused by the high temperature, must also sustain the weight of from 12 to 20 tons of molten metal.

The ash-pit is very advantageously provided with iron doors, which may be closed during the ladling, to exclude all currents of air, while the flue is brought as nearly as possible over the skimming-door, in order that the air-current that enters therefrom may ascend at once without affecting the metallic bath.

The two bottoms are smelted in with unusual care, and the upper one thoroughly saturated with repeated small charges of metallic copper. This should be spread over the entire surface in the shape of granules, and should be rapidly fused until it is en-

* A few unimportant exceptions to this statement may still exist.

tirely liquid. At first, nearly all will be absorbed, but eventually, a larger and larger proportion will be regained, and thoroughly dipped from the ladling-hole at the close of each operation. A hearth of the ordinary size, 9½ feet by 14 feet, will absorb from 6,000 to 18,000 pounds of copper during the "soaking" process, according to the quality of the sand and the temperature attained during the "smelting in" of the bottom.

On no account should metal of poor quality be used for purposes of saturation. This is a fatal economy, as the grade of all copper refined in the furnace for many months may be affected thereby.

Even after the bottom is well saturated and has attained a considerable degree of firmness, the careful refiner will avoid any possible injury thereto from heavy masses of metal. It is not uncommon to charge pigs of blister copper weighing 1,000 pounds or more, the sharp corners and edges of which are very likely to cause indentations and unevenness in the toughest bottom, which may serve as the starting-point for serious after-effects. All such injuries may be avoided by laying down a rough floor of old planks or similar material.

The fuel best suited to refining is a not too caking bituminous coal with long flame. Sulphur-bearing coals should be avoided, as tending to alter the "pitch" of the metal at critical moments. Where such coal is expensive, a cheaper variety may be used for the earlier stages of the process.

No better fuel exists for refining than wood, as its freedom from sulphur and other impurities, and the long, pure, non-reducing flame that it yields, peculiarly fit it for the purpose. It was used entirely at the Ore Knob Refining Works with great satisfaction, and, were it cheaper, would be more frequently employed at the great copper centers.

The size of the charge is limited rather by custom and the capacity of the attendants than by the size of the furnace, and has been greatly increased of late years.

During the writer's student years a charge of 14,000 pounds was considered large, but the present English refiners vary from 18,000 to 30,000 pounds and upward.

The Lake Superior refineries are charged with some 30,000 pounds of 80 per cent. "mineral," producing over 20,000 pounds of pure copper. The principal trouble with large charges is the tendency of the refined copper to get "out of pitch," when retained in a molten condition for so long a time that fresh coal must be several times thrown upon the grate. Another difficulty is the want of room between hearth and roof to accommodate such a weight of metal. The shape and irregularity of the pigs of blister copper, and the difficulty of accurately placing such awkward bodies in the desired position in a red-hot furnace, have also prevented the ordinary use of larger charges. This is remedied in the Lake furnaces by lessening the customary pitch of the roof from bridge to charging-door, and curving it down abruptly at the latter point.

Great care must be observed regarding the quality of all material allowed to enter the refining-furnace. It is not an apparatus for the concentration of matte, but simply to alter the shape of metal that is already nearly pure, and to put the finishing touches on it. Much of the pig-copper produced from blast-furnace work from both carbonate and sulphide ores may advantageously undergo a preliminary purifying process in the blister furnace. All copper below 96 per cent. should be thus treated; a mere melting down with free admission of air being sufficient to produce a 99 per cent. blister copper in most cases, so that two moderate charges can be thus treated in 24 hours.

A few hundred pounds of the richest refinery slag from the last skimming may be returned to the same operation, the rest going back to the last preceding operation.

Cement copper from wet processes should, in most cases, be treated in the blister furnace. It must be thoroughly dampened to prevent mechanical loss, and when mixed with white metal to the extent of one-fourth or one-third of the entire charge, assists so materially in enriching the product and in shortening the operation that it just about repays the cost of its treatment.

Mr. James Douglas, Jr., has regularly produced so pure cement copper, by both the old and new Hunt & Douglas process, that it is refined at once with advantage. The principal drawback is its excessive bulk, which renders it necessary to add the cement copper in several successive portions, to obtain a full charge. This may be obviated by pressing it into bricks while yet damp.

Only one charge can be treated in a refining-furnace each 24 hours, and in ordinary cases, the labor connected therewith consists of one head refiner, three or four ladlers (according to whether the refiner acts also as ladler), one night refiner, one man to lift the ingots from the boshes, one to dump the molds, and one to remove any accidental impurities from the ladles, dry the molds, etc. The latter three operations are often conducted by boys.

One man is also required to remove the ingots to the packing-house, while the packing itself, and the transportation and manipulation of the new charge, are effected by the furnace *personnel*, which usually expects to conclude the day's labor by 3 P.M. The work is very hot and severe while it lasts, and in the cases of the large charges referred to, extra assistance may be required for packing the copper and similar extraneous work.

While the quality of the copper depends largely upon the skill of the refiner, its external appearance and neatness are principally influenced by the ladlers. As these latter qualities exercise an undue influence upon the sale of copper in this country, it is of great importance to create a body of trained and skillful workmen, whose pride, as well as self-interest, is enlisted in the matter.

The color of the copper has an influence with American buyers entirely disproportionate to its importance as a sign of purity. A deep rose-red is the color most prized, while any brassy appearance is very damaging in the eye of the buyer.

As this dirty yellow appearance can be produced at pleasure by allowing the copper to remain a few seconds too long in the molds before it is dumped into water, while the poorest copper may be colored a fine, deep red by lifting it out of the water for a second immediately after dumping, and then returning it again to the trough, as well as by the use of any one of innumerable baths or "pickles," it certainly should not be regarded as of such vital importance. It is true, nevertheless, that *pure* coppers take on the desired color with much greater ease than those containing arsenic, antimony, or various other substances, and in the case of the remarkably pure Lake Superior metal, it is even difficult to produce that dreaded brassy appearance which any impurity of water or want of care is certain to develop in ordinary cases.

The red color is produced by the formation of a minute film of suboxide of copper; but why this hue should be affected by slightly brackish water, or by changes of temperature in the cooling water, it is difficult to understand. Pure water should be used and the most favorable temperature discovered for each variety of copper. In some cases, it must be nearly boiling; in others, ice-cold; while in still other instances, the refiner corrects an unfavorable coloring by the introduction into the cooling boshes of soda-ash, salt, sawdust, coal ashes, and various other apparently inactive substances.

The refining-furnace usually receives its fresh charge immediately after it has been emptied of the preceding one. The fire is not urged until evening, in order that the first two stages of the operation, the *fusion* and the *refining*, may not be completed before the day shift comes on to execute the *refining* proper—the third and final stage.

With pig-copper of reasonably good quality, the process of fusion may be begun at 7 or 8 P.M., and be pushed as rapidly as possible. Owing to the high heat-conducting quality of this metal, the pigs retain their shape until the fusing-point is reached, when they soften and melt almost instantaneously. From the reducing character of the flame, only slight chemical changes have thus far been produced; but as soon as the protecting layer of slag is removed from the surface of the bath, and air freely admitted, the process of purification proceeds with great rapidity, from the direct oxidation of the foreign substances present, as well as the more far-reaching and powerful reaction of oxide of copper upon all those metalloids and bases that have a greater affinity for oxygen than the copper itself. A thin slag forms rapidly upon the surface, and is removed at intervals of an hour or so. The constant escape of anhydrous sulphuric acid causes a persistent ebullition, which tends greatly to facilitate the process of oxidation. As the proportion of base metals becomes diminished, the slag is more strongly colored with the red oxide of copper, until that produced toward the close of this stage contains from 40 to 70 per cent. of this metal, and becomes a valuable oxidizing flux for the preceding blister process. The total amount of slag produced during the operation of refining depends principally upon the quality of the pig-copper, but is seldom less than 12 per cent. of the entire charge, containing from 4 to 6 per cent. of the total weight of copper.

The gradual cessation of ebullition and the rapid formation of oxide of copper by no means indicate the entire disappearance of the sulphur present, which, from its strong affinity to copper, remains dissolved in the bath with great tenacity. If the oxidizing process has been sufficiently thorough to insure the presence in the liquid metal of a perceptible quantity of suboxide of copper (from 0.2 to 0.7 per cent., according to different authorities), a small sample ingot poured at this stage will exhibit a very peculiar and characteristic phenomenon. On cooling, it will suddenly rise in a line along the center, often forming an abrupt ridge several lines in height, and having an irregular and granular fracture. This is said to be due to the absorption of sulphurous acid, a property only possessed by metal containing a considerable proportion of suboxide of copper, but still unrefined and tenaciously holding on to a trace of sulphur and other impurities. The process of "flapping" or "rabbling" is now begun, by which the liquid bath, through the side door, is constantly agitated in a peculiar manner by means of a small rabble.

It is, of course, a purely oxidizing operation, both tedious and slow, requiring, on an average, two hours of constant work. Although seemingly a most awkward and ineffectual means of agitating an extensive bath of molten metal, and bringing all its particles in contact with the atmospheric air, it has never been improved upon. The copper now becomes "dry" from the dissolved suboxide, and when poured into a mold, sets with a deep depression upon its surface, while its fracture has a characteristic mottled appearance, following upon a previous fine-grained surface, as particularly mentioned by Professor Egleston in his valuable paper on "*Copper Refining in the United States.*" The color is a brick-red, but both grain and color are so influenced by the temperature at which the metal is poured, as well as by the rate of cooling as determined by the size of the test-ingot, that these signs must always be taken in conjunction with other and more reliable indications. The metal during this period is undergoing a powerful scorification from the dissolved oxide of copper, and most injurious impurities are gradually oxidized, and either effectually removed by slagging or volatilization. Certain metalloids, however, resist this scorifying influence to a remarkable degree, and consequently have a most injurious effect upon the refined metal. These are arsenic, antimony, and tellurium, mentioned in the order of their frequency. The extreme importance of the subject warrants the mentioning of the best means to remove the two first-mentioned impurities, the latter having come but once within the author's experience, and probably requiring the employment of one of the electric or chemical methods, by which excellent copper can be made from very poor material.

A careful trial of Vivian's invention of dry-sweating, by which the impure blister copper is exposed to a long oxidizing heating just below the fusion-point, has not succeeded with the writer; but the addition of from 3 to 5 per cent. of pure white metal—subsulphide of copper—to the bath at the beginning of the refining process (as suggested by some person forgotten by the author) has a most rapid and satisfactory effect in removing both arsenic and antimony. Very bad cases may require two such additions, with an intervening oxidizing operation. A still more sure and radical method consists in exposing the arsenical ore to a dead roast, and subsequently smelting the same with a large proportion of iron pyrites—cupriferous, if possible. The resulting low-grade matte should be regarded and treated as a sulphide ore, and will, if the initial calcination is thoroughly conducted, be free from either arsenic or antimony.*

The process of reduction follows that of oxidation and the sub-oxide of copper, having served its purpose as a purifying agent must now be reduced to metal again; otherwise, the copper would be brittle both when cold or at higher temperatures, and unfit for manufacturing purposes. The reduction is effected by means of a long pole, as large as can be introduced into the furnace and of any kind of green wood—hard wood being the most economical. This, being buried in the metal bath, evolves an immense volume of hydrocarbons and other reducing gases, and rapidly removes the excess of oxygen. The surface of the metal is also covered with charcoal, to prevent access of air, and samples are constantly taken to determine the condition of the copper. The entire removal of all the oxygen present is impossible, even overpoled copper, according to Egleston,† containing over 0.1 per cent. of oxygen. An otherwise tough copper may become brittle from overpoling, and this is doubtless due to the fact that the impurities that were present in the tough copper were dissolved as oxides and consequently innocuous, but on being reduced to the metallic state at once asserted their deleterious influence.

The poling usually lasts an hour or more, and is continued until a full-sized test-ingot shows no contraction or depression on cooling, and the texture is extremely fibrous and silky, and of a beautiful rose-red. Further tests are made by nicking and bending test-bars, and by hammering out a piece into a thin plate, which should show no cracks at the edge. This condition of tough-pitch

* The author is unable to give the original sources of many statements here made and tested by himself with satisfaction, and desires to distinctly disclaim any originality in any operation or apparatus pertaining to copper metallurgy; having always preferred to adopt those improvements that have been thoroughly tested by others of a more original turn of mind.

† See *Copper Refining of Lake Superior*, by T. Egleston.

is essential to copper used for rolling or wire drawing, but is entirely superfluous for ingot copper that is to be used for brass founding, as it may be easily imagined that the fusion that it undergoes in the brass-founder's crucible under various oxidizing and reducing influences, effectually upsets the exquisite niceties of the refining process, so far as the proportion of dissolved suboxide is concerned.

A volume could be easily filled with practical comments upon

Fig.5

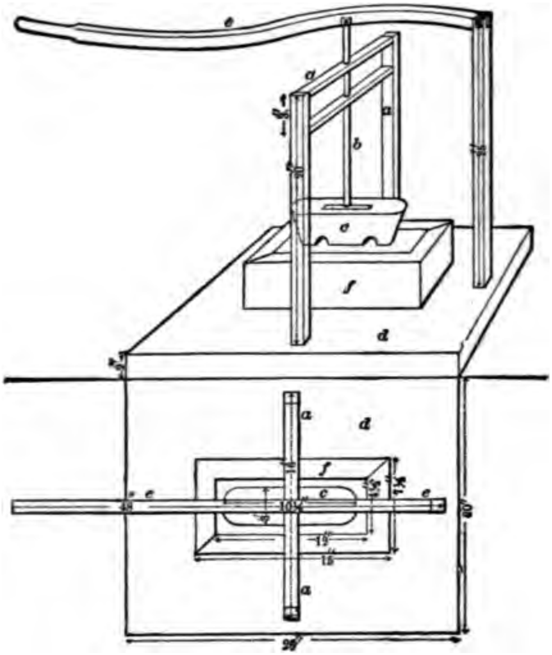


FIG. 68.—STAMP FOR INGOT MOLD.

the process of refining, but space forbids any further details. The addition of lead to copper intended for rolling is quite common in England, and is doubtless beneficial to many impure coppers. The purer copper of the Lake district and from the Arizona carbonates does not seem to receive any benefit from this practice.

The molds used for the casting of ingots should always be made of copper, and are easily and rapidly produced by the ordinary ingot stamp, as illustrated herewith. The proper taper of the mold and the proportion of surface in contact with the ingot have an important effect upon the ease with which the mold delivers. When the copper is ladled too hot, the molds are rapidly ruined, and as at best they wear rapidly, they should be returned to the refining furnace as fast as they become in the least imperfect; otherwise constant annoyance and accidents will result from the obstinate sticking of the ingots.

The Ansonia Brass and Copper Company has patented a mold for casting ingots directly from the furnace, without the intervening process of ladling. While such an improvement would relieve the workmen from the hottest and most laborious portion of the operation, the very nature of the metal, its high fusion-point and great heat-conducting capacity, cause it to chill so suddenly as to render the success of such an invention a matter of some doubt. The same company employs a gas generator for heating a single refining-furnace, and although pronounced convenient and successful, it can hardly make any great saving, considering the small amount of fuel generally used in ordinary refining, and the great expense of the generator plant.

A great saving in the expense of refining has already been made by increasing the capacity of the ordinary furnace, and the next important improvement may be looked for in bettering the quality of the refined copper and increasing its strength and tenacity. How this is to be effected is far too difficult a subject to be discussed within the limits of a practical paper on existing methods. Experiments conducted by Mr. Patch, of the Detroit Copper Company, as well as the writer's personal trials, seem to indicate that the presence of suboxide of copper is by no means essential to the greatest malleability and strength, as believed by Percy, and that a proper method of treatment may result in the production of copper having a strength far beyond the best brands at present known.

The cost of refining varies so greatly with the purity of the blister copper treated, and depends also so completely upon the size of the charge, that no absolute estimate of expense can be given.

The following figures, taken from actual practice, give a fair idea of the cost of refining ordinary Arizona pig-copper of from 95 to 98 per cent.—being about equivalent to good Chili bars. The size of the charge is assumed to be sufficient to produce 24,000 pounds of refined metal, the furnace running regularly, and mak-

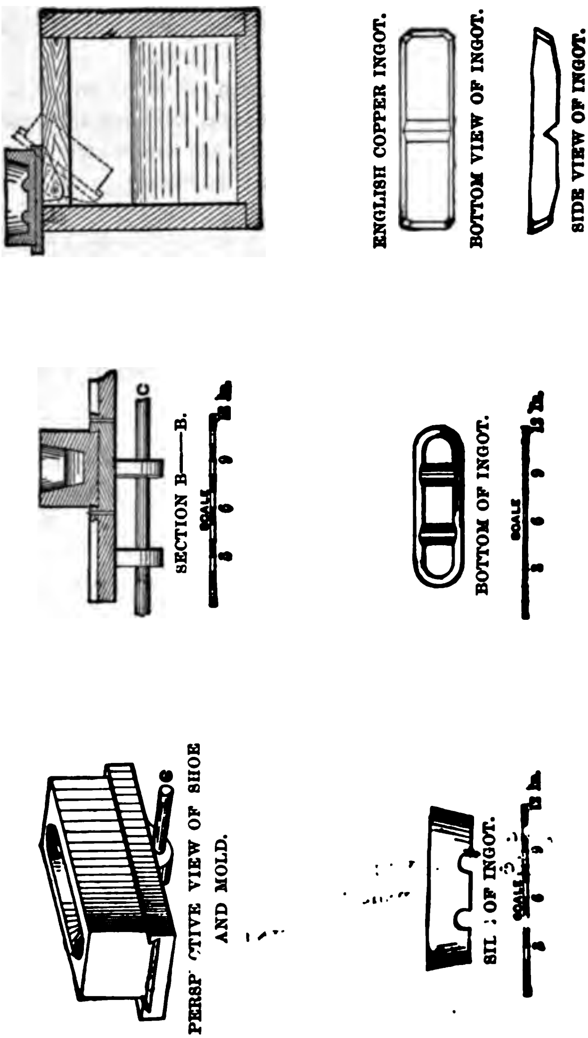


FIG. 69.—DETAILS OF INGOT, INGOT MOLD, SHOE AND TANK.

ing one charge every 24 hours, while the expense of foremen, etc., is supposed to be divided between two furnaces.

Cost of refining one charge, yielding 24,000 pounds of copper—

Coal—best quality—3.8 tons at $5.50	$20.90
Clay and sand for fettling—450 pounds	.95
Cheap clay and loam for doors and slag-beds—400 pounds.	.30
Poles—45 feet of 6-inch poles at 4 cents	1.80
Charcoal—6 bushels at 10 cents	.60
Proportion of cost of renewing bottom	.34
" " " main arch	.62
" " " flue	.32
" " other repairs on furnace	.72
For renewing tools, barrows, ladles, etc	1.11
Repairs on " " " "	.64
Lights, oil, soap, clay-wash, brushes, etc	.70
Cost of resmelting poorer slag in blister-furnace	1.20
One head refiner	4.00
One night refiner	3.00
Four ladlers at $2.75	11.00
Man fishing ingots	1.50
Boy dumping molds	.75
Boy removing specks from ladles while pouring	.75
Man wheeling copper to packing-room	1.50
One laborer about furnace	1.50
One head packer	2.50
Two assistants at $1.50	3.00
Miscellaneous expenses of packing, paint, stencils, etc.	.65
Cost of pumping water for boshes	1.15
Proportion of day and night foreman	3.00
Proportion of expense during Sundays and other delays	1.44
Proportion of assaying necessary for control of operation	1.12
Grand total	$67.06

Cost per pound 0.2794 cents.

This agrees closely with the actual cost of running large refining works where prices closely approximated those assumed in this estimate, being just three-tenths of a cent, including general expenses.

GAS-FIRED REVERBERATORIES—REFINING COPPER WITH GAS IN SWEDEN.

At Atvidaberg,* in Sweden, a regenerative gas plant is used to heat the small reverberatory furnace in which the cement copper,

* Mr. Paul Johnson, of Sweden, an experienced copper metallurgist, has kindly furnished this description of work at Atvidaberg. This has subsequently been revised by Mr. Carl Rudelius, the manager of the Atvidaberg works.

obtained from cupriferous pyrites by the use of the Henderson extraction method, is first made into blister copper, and eventually refined upon the same hearth, in a second gas-heated furnace.

The small scale of working here practised cannot be recommended, as it is not conducive to economy. But the successful employment of a wood-burning gas-producer in copper-refining is so interesting and instructive, and bears so strongly upon the conditions existing in many parts of our own country that I think it well worth while to embrace the opportunity that has presented itself of describing it minutely, and with detailed working drawings to illustrate the apparatus employed.

The cement copper is charged in a moist state, containing 50 to 60 per cent. copper, with refinery slags and a small amount of roasted, high-grade matte, and a considerable proportion of charcoal, to prevent the oxidation of the fine particles of metallic copper. The produced blister now averages 99 per cent. copper.

The gas regenerator is fired entirely with a very cheap and poor quality of fuel, consisting of refuse wood from building operations, ends of scantling, slabs, limbs of trees made into fagots, and similar waste material, and the gas is purified from tar by direct contact with water.

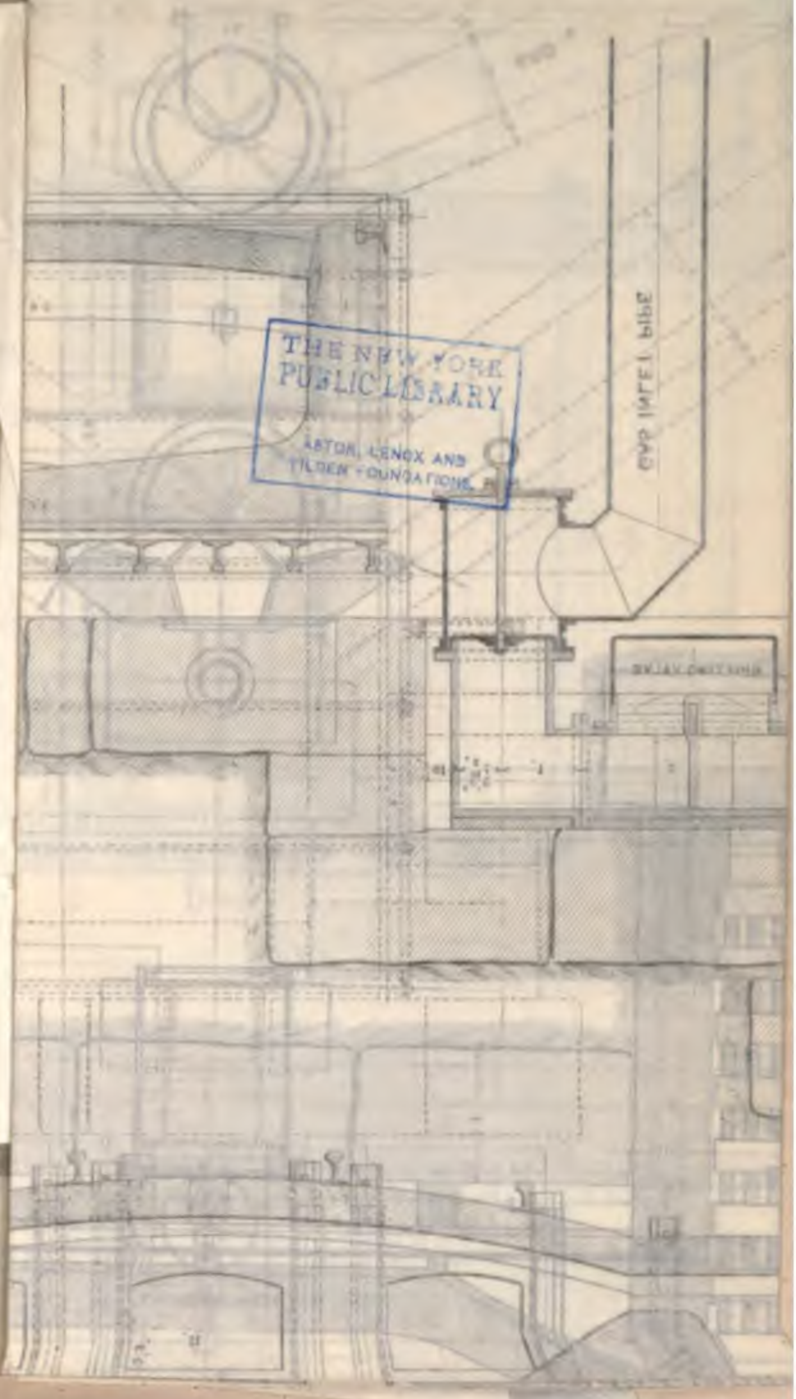
THE BLISTER PROCESS.

The operations here described were executed in the year 1889, and starting with a cold gas-producer, and furnace hearth, the latter merely having been slightly warmed by a small wood fire upon the hearth. Plates XVIII. and XIX. show this furnace and its gas-generators.

This small fire was kept upon the hearth about sixteen hours before filling up the gas-producers, the flame being allowed to pass out through the regenerators on either side.

At the expiration of this time, and at about 6 A.M., the fire in the gas-generator was kindled, the latter within the next three hours being filled nearly to the top with the refuse wood already mentioned.

By 10 A.M., four hours after the kindling of the fire in the generator, it was considered safe to let the gas into the furnace, it having hitherto passed off into the air. Any carelessness in letting the gas into the furnace before all air is driven out of the generator might result in a serious explosion. The gas is shifted from one regenerator to the other every 10 minutes by means of a



valve, and by 6:30 P.M., 12½ hours from starting, the hearth of the furnace had reached a bright yellow heat.

The bridge-wall and other exposed parts were then repaired in the customary manner by tamping into the burnt-out cavities crushed quartz, sieved through a screen with eight meshes to the linear inch, and at 8:15 P.M., 14¼ hours from the start, the furnace was hot enough to begin charging.

Nine cubic feet of charcoal fines (a totally valueless material where charcoal is used, and that can be obtained without cost) was first dumped upon the furnace-hearth, then 300 pounds of thoroughly roasted copper matte, containing about 50 per cent. copper, then 600 pounds of refinery slags, containing 40 to 60 per cent. copper, then some two cubic feet of charcoal again; on top of this came 5,000 pounds of cement copper, and last of all 300 pounds more of roasted matte, after which the doors were luted and the gas turned on again.

The great bulkiness of the cement copper requires that, in this small furnace, the charging should be done in two separate operations. At 9:45 A.M., 13¼ hours after charging, the material was sufficiently fused down to admit the addition of the remainder of the charge. Four cubic feet of charcoal fines were first thrown upon the half-molten bath, then 3,000 pounds cement copper, then 300 pounds each of roasted matte and refinery slags, and, last of all, about 40 pounds of dirty cement copper, from the straw filters of the precipitating tanks.

The working-door was luted up at 10:05 A.M., and at 2 P.M., after four hours' firing, the charge was sufficiently fused to admit of the first slag-skimming, which was conducted in the customary manner. At 3:30 and 4:45 P.M., slag was again skimmed in small quantities, the charge still being far from liquid all through, and much of it still sticking to the hearth.

As the blister copper is here tapped into iron molds (an excellent plan which cannot be too highly recommended to our own smelters for its cleanliness and economy), these are now heated with slag from a fourth skimming at 5:45 P.M., at which time it was found that the bottom was getting clean, and that bubbles were beginning to rise through the molten mass, showing that the chemical reactions between the sulphides and oxides present, as well as between the carbon (charcoal) and oxides were rapidly progressing. At 6:20 P.M., a considerable quantity of tolerably liquid slag was skimmed, and a thorough stirring of the bath with a stout iron hook took place, to free the bottom and assist in breaking up the still unfused masses. A slight blast was now forced through openings in the fire-bridge to cool the overheated brick-work in that portion of the furnace, and at 6:40 P.M. slag was again skimmed. At 7 P.M. the copper was skimmed tolerably clean of slag, though a considerable film of matte still floated upon it, and the blast was now let on through two blow-holes in the rear of the furnace to hasten the oxidation of the sulphur and iron. Sometimes, instead of using blow-holes, a tuyere is introduced into a small door opposite the charging-door, and there luted in place.

At the same time, the admission of air to the gas-producer was increased so as to make the flame more oxidizing, and permit it to play more freely over the whole extent of the bath.

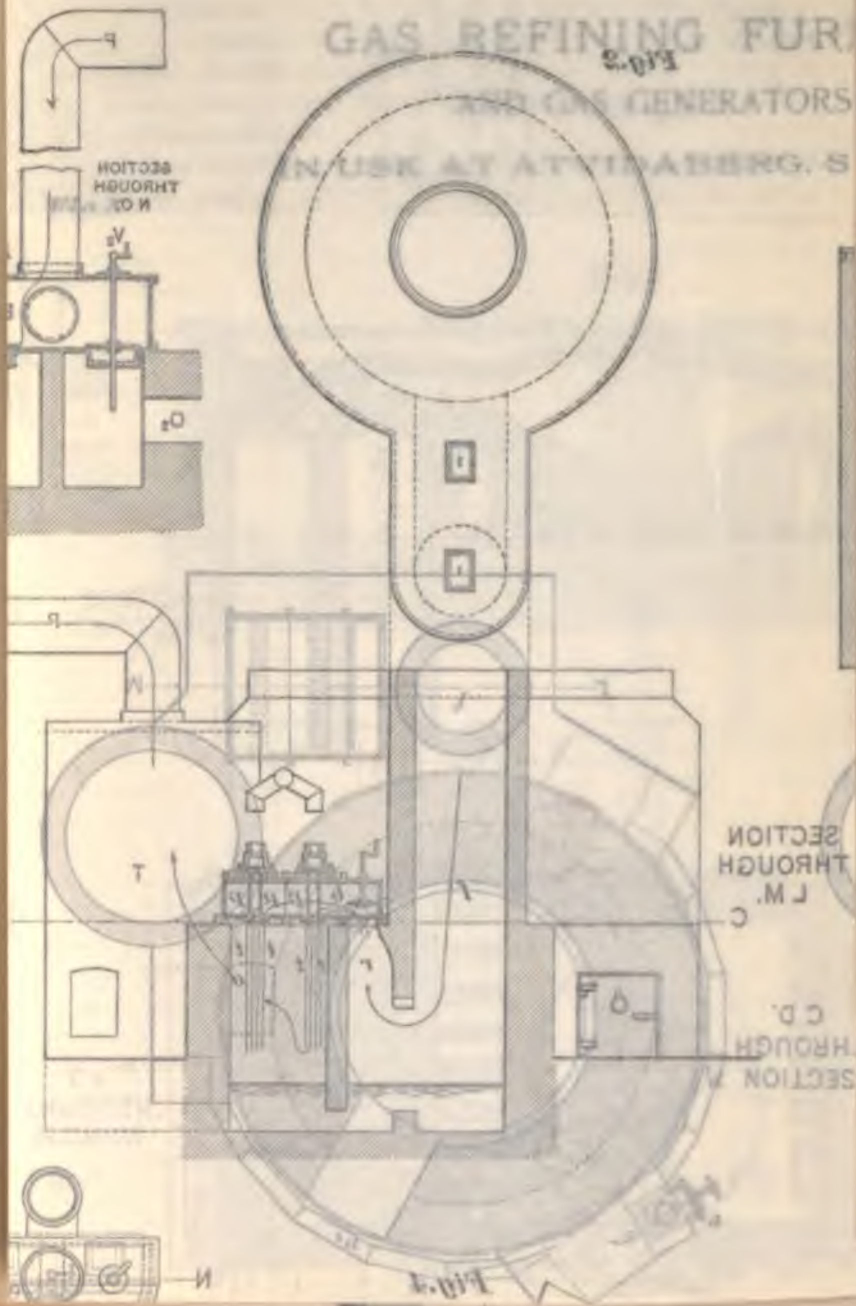
Between 7 and 9:20 P.M., four skimmings were made, and at the last one, the bath was again skimmed clean, and a test taken in a small ladle, which showed a pretty even surface.

The sulphurous acid gas is escaping rapidly in a multitude of fine bubbles (German "Braten"), and this action increases in violence until globules of molten copper are thrown out through the working-door. The slag, which at first was dense, now becomes pasty and full of bubbles.

Tests taken every few minutes showed a disposition to rapidly become thinner and more concave, until at 10:10 P.M. they assumed their proper form, *i.e.*, swelled up with a hollow in the center like a baker's roll, the bottom portion, when broken, showing a reticulated surface.

The blister copper is now tapped at once, after a few moments of strong heating, into iron molds, formed on the bottom by a large iron plate, while the sides are made of strips of iron shaped like a triangular gutter, and laid on their open side, the apex of the triangle being uppermost. Small cross-divisions of sand are made between these parallel sides, and the sand is covered with a thin layer of clay mortar to strengthen them, and which must be thoroughly dried before the metal is tapped into them.

The tapping of the blister copper is accompanied by a strong evolution of sulphurous and sulphuric-acid gases, and requires some skill in its management, being guided and controlled in its impetuous course by long wooden poles in the hands of the furnacemen. As in the common English and American practice, the pigs are broken apart at their junction while still almost white-hot, and as soon as the outside of the pig has become strong enough to


GAS REFINING FUR
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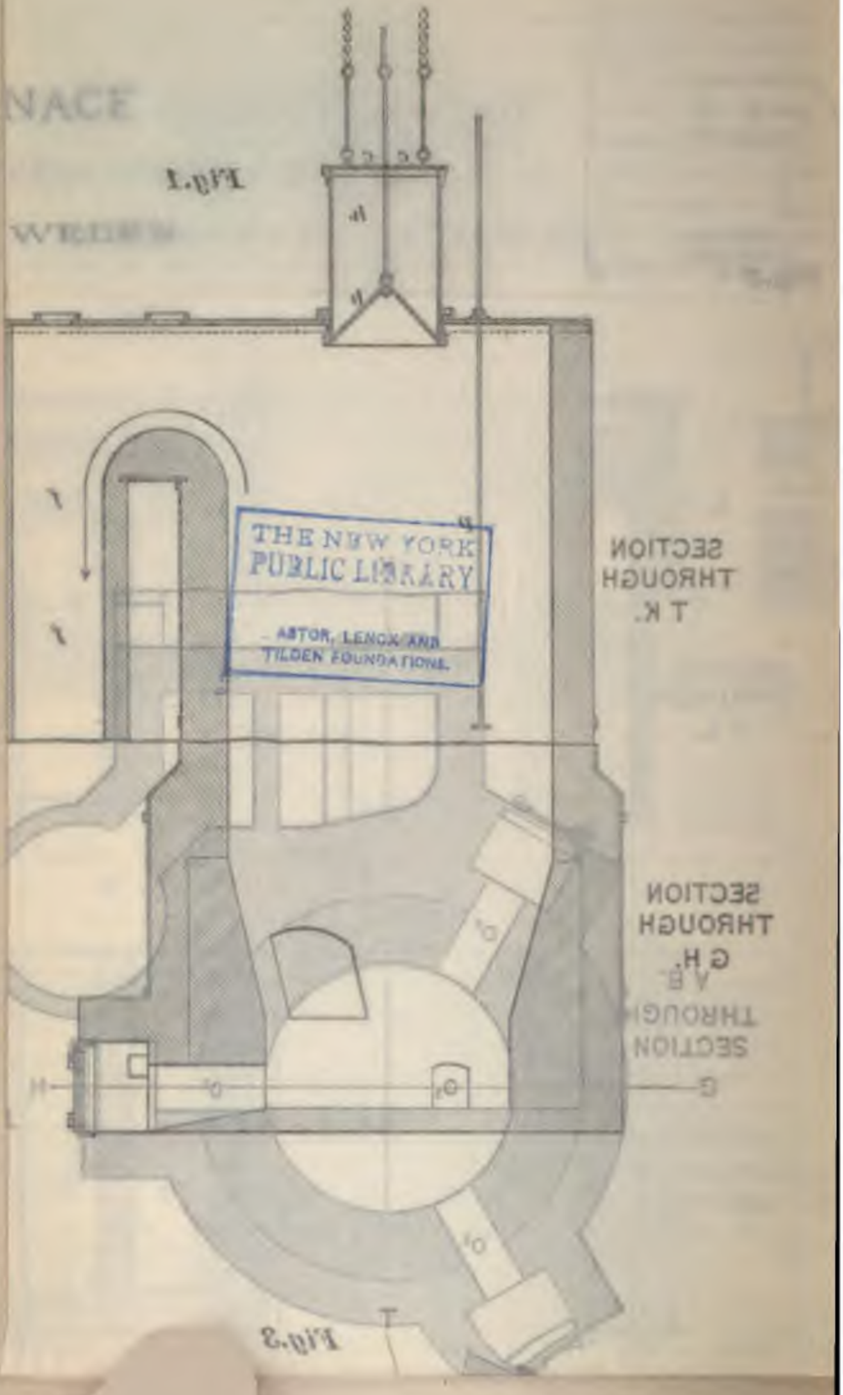
SECTION
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G H.
Fig. 3

allow it to be pried upon with a heavy iron bar. At this stage, the blister copper is very red-short and brittle, but if the critical moment is neglected, it becomes tough and unmanageable, necessitating a most laborious and expensive recourse to cutting apart with chisels. A little sand or fine charcoal is thrown upon the necks of the pigs to keep them hot until the pig has solidified enough to bear handling.

The furnace is now scraped out clean, and the sides and bridge mended with crushed quartz in the usual manner, a long plug of wood being first driven into the tap-hole, and thoroughly packed with damp sand, so that it soon carbonizes, and becomes a plug of charcoal which is easily penetrated by the tapping-bar. On this small furnace, about 140 pounds of quartz are used for repairs after each charge, and this is heated strongly for about an hour.

Only one charge is made in twenty-four hours, the actual time taken for the operation being about 21½ hours, and the furnace-gang consisting of one head smelter, one assistant, and one man at the generator, all on eight-hour shifts.

The consumption of fuel in this operation is very difficult to estimate, as every scrap of waste wood is used, in the shape of old roots, branches, ends of sawn-off timbers, etc.; also sawdust. About 850 cubic feet of waste limbs and 400 cubic feet of waste wood were used in 24 hours, but as this was all estimated by measure, and as from its irregular shape the interstices bore an enormous proportion to the amount of solid wood employed, it is evident that a very reasonable consumption of fuel took place.

The taking of tests from the bath, and the judging of the forwardness of the operation therefrom, are difficult and delicate, and depend largely upon experience. As the blister copper in our large blister-furnaces presents exactly the same series of changes, and as inexperienced men are constantly perplexed and mystified by this matter of tests, I will give a brief description of the appearance of these tests at Atvidaberg, as the same description will apply equally well to our own conditions.

The tests are taken at most works in a small ladle about 1½ inches in diameter by ½ inch deep.

A test taken early in the operation, and before the process of oxidation has advanced very far, is level at surface, and shows a dull gray fracture, with few shining points.

As the oxidation of the sulphur progresses, the assay seems to spread out thinner, and on fracture shows less dullness, and a multitude of small bubbles begins to appear.

Still further on, the bubbles begin to enlarge, becoming the size of small peas, while the surface of the test rises and becomes more convex. This increases till it looks very much like a baker's roll, and shows an indescribable vitreous luster in the interior of the bubbles on fracture, when the blister process is complete and the copper is ready to tap. If the oxidation were carried further, there would not only be a heavy loss of copper in the slag (there is a much less loss in the refining operation later, as the slow melting-down of the blister copper assists greatly in oxidizing the last traces of impurities without much slagging of copper), but the blister copper would lose is valuable quantity of excessive "red-shortness" by which we are enabled to easily break the pigs apart, and other practical disadvantages would occur.

But if, in spite of these drawbacks, the oxidation of the copper is continued, the surface of the test becomes very convex, and finally it bursts, and, as the smelters say, "the worm crawls out," by which they mean that the test "spits," or has an outgrowth of molten copper on the surface, which looks like a worm, and is identical with the common "spitting" of a large silver button when cooled too quickly after cupellation.

Of course, as soon as the impurities in the blister copper are thoroughly oxidized, and indeed long before this point, the copper itself begins to oxidize rapidly, and to form a silicate of copper, taking up quartz from the hearth-sides and fettling. I remember one instance in my early practice, when I had engaged a new refiner, and was called away just toward the close of the oxidizing stage, but anticipated no trouble, as the man brought good recommendations. When I returned in five or six hours, I was informed that the copper was all sinking away and going to slag, as was indeed the case, the bath in the furnace having shrunk to infinitesimal dimensions, while the whole shed was filled up with blood-red slag. It seems that the new refiner had kept on oxidizing, expecting to see some peculiar appearance of the test which he was used to in refining the very impure copper that he had always worked on. Our copper being from a wet process, and of the highest grade of purity to begin with, failed to show this peculiar phenomenon, which he termed "The bloomin' o' the Butt," if any one knows what that means, and he had patiently kept on oxidizing, regardless of the ominous lessening of the bath. The slag was all put back, with some charcoal and a little high-grade matte, which furnishes the best reducing agent that we have in the reverberatory furnace, and in the course of a few hours the copper was again obtained, and this time in a proper state of purity.

THE REFINING OF THE BLISTER COPPER IN GAS FURNACES AT ATVIDABERG, SWEDEN.

The operation of refining at Atvidaberg is conducted in the small furnace shown on Plate XX., Figs. 7 and 8.

The charge of blister copper is very small, only about 5,000 pounds, and in a furnace in normal condition this is fused down in about four hours. The bath is then skimmed clean, the amount of slag being very small, owing to the unusual purity of the blister copper.

A light blast is turned upon the surface of the molten metal by means of a small tuyere introduced through a door opposite the working-door, and the copper is at the same time "flapped" constantly with a small rabble in the usual way. Of course the object of this stage is to expose the molten copper to the action of the air as rapidly as possible and thus promote quick oxidation of the impurities that it contains, for the whole process of copper refining is based upon the fact that the impurities that are commonly contained in copper oxidize more easily and quickly than the copper itself. If they happen to possess a lesser chemical affinity for oxygen than the copper itself, they will, of course, remain unoxidized. A familiar example of such unoxidizable substances are silver and gold, which remain alloyed with the copper throughout the refining process, and indeed could, theoretically, be separated from the copper by oxidizing and slagging it off, leaving the precious metals behind, as in the cupellation of lead. In practice, this method has not been found economical, except under certain rare conditions, where it plays an important part in the separation of these metals.

I doubt very much if the air-blast from the tuyere striking the surface of the copper produces any very marked acceleration of the oxidizing process. I have more than once tried it under similar conditions in the ordinary English method of refining as practised in this country, but could not see that it made any perceptible difference, and am inclined to think that it is merely a relic of the old-fashioned German refining method, where the tuyere was necessary to produce an air-current on the surface of the molten bath, there being no arch over the German refining hearth, and consequently no strong, natural air-current over the surface of the metal as there is where a reverberatory furnace is used.

But the degree of energy and skill with which the "flapping" is executed has a very marked effect upon the time necessary for this stage of the operation. It is most difficult, violent, and exhausting labor, and requires much tact and experience to strike the bath with the proper force and just at the proper depth to break it at that point into a shower of drops, which fall over a large area, and at the same time send out a large number of concentric wavelets of copper, that present a greatly increased surface to the action of the air-current that is hurrying over the surface of the bath to gain the chimney.

At Atvidaberg, this stage of the process usually occupies $1\frac{1}{4}$ hours, at the end of which period a test taken in the small ladle should show a very coarse, crystalline, light-red fracture, the small bubble in the center being a bluish-black instead of the reddish-brown hue that it showed during the early part of the oxidizing period.

The impurities are now nearly all removed, or else exist in the copper in the shape of oxide, while the metallic copper probably contains a considerable amount of suboxide of copper in solution.

The bath is skimmed clean, and the "poling" now begins, in order to lessen the amount of suboxide of copper (with which it was necessary to saturate the metal in order to thoroughly oxidize all foreign elements), as well as to reduce the foreign oxides dissolved in the copper. As these are reduced, they act in three different ways. Such as are at all volatile pass off in gas. A considerable portion of the fixed oxides is taken up by the slag formed during the operation. While a third, and very small portion, is reduced back to metal and remains in the copper as impurities, iron being the principal of these.

The poling operation is divided here, as elsewhere, into two stages:

Dense poling and

Tough poling.

A large birch pole is thrust into the working-door, its butt being forced down into the molten bath by forcibly raising its thinner end which projects into the air, using the upper edge of the furnace door as the fulcrum, while the pole is retained in position by a piece of notched plank, set under its projecting end. The ebullition of gases is very great, and a rapid reduction takes place in the copper, while currents are established in every direction, so that all the metal in the furnace is rapidly brought under the influence of this powerful reaction.

The crystalline texture of the fractured test becomes of a much finer pattern, and the color much lighter and more pinkish, while the bubble in the center becomes rapidly smaller and more yellow, and soon disappears. When these reactions are fairly established, and the test is seen to take on a dense, fibrous texture from below upward, the stage of "dense poling" is considered at an end.

It usually takes about forty-five minutes, during which time the slag is skimmed several times, and as it now largely consists of a subsilicate of the oxide of copper, which is too thin to skim off, coke dust is thrown on the surface of the bath to thicken the slag.

At the beginning of the "tough poling," a barrel of clean, selected charcoal is thrown upon the clean surface of the bath, and a fresh pole introduced. Tests are now constantly taken, the changes in the quality of the copper, though chemically very slight, being now both physically and commercially very important, and succeeding each other with extraordinary rapidity.

The fracture of the test is now becoming pinkish and fibrous, and minute pearly globules begin to appear under the magnifier, while the toughness of the metal increases in a remarkable manner.

As the process progresses, the even rows of fibers begin to disappear, and an irregular, very finely grained fiber replaces them, while the fracture looks more silky and the toughness increases.

In two or three minutes from this time, the pearls suddenly become distinct, the beautiful rosy pink of refined copper appears, and the test becomes so tough that, after nicking it with a chisel, it can be bent back and forth many times in a vise without breaking.

The pole is now withdrawn, fresh charcoal is spread over the bath to entirely exclude the air, and the heated and clay-washed ladles are brought to the door.

The ladling of this small charge of 5,000 pounds or less took over an hour, having to be interrupted several times to pole the copper, and well exemplifying the claim I have so often made for large furnaces and large charges, that it is infinitely more difficult to handle small quantities of metal, as the slightest mischance, or undue splashing about of the copper by the ladles, causes oxidation, and gets it "out of set."

When we are ladling a proper amount, say 25,000 to 35,000 pounds of copper in one operation, we seldom have to put in a pole more than once or twice until we come nearly to the end of the ladling, and are dealing with something like the quantity that they begin with in Sweden.

And in order to be sure that there shall be no mistake in the quality of the copper, I have usually made a practice of closing the operation as soon as it becomes difficult to keep the metal "in set," and ladling out the little copper that remains in the furnace into sand-beds, to be thrown back into the furnace at the next refining. There is no waste in this, and practically no expense, the sole drawback being the couple of hundred dollars invested in the extra amount of copper that is thus kept circulating in the process, and gets into market 24 hours later. With interest at 10 per cent. per annum, the cost of this practice would be about one-half cent per day, and allowing even 10 cents for the amount of fuel necessary to resmelt it (for the labor costs nothing, it being all done under contract, and the extra labor of throwing a few ingots back into the furnace not being heeded by the men), it will be seen that the cost is not to be considered in comparison with the very serious trouble and expense arising from sending out copper of unequal quality.

I mention these details, as I have heard this practice denounced as wasteful and extravagant, while I consider it a decided improvement, if a small one.

GAS FURNACES IN AMERICA.

Apart from the gas-fired refining-furnaces that have long been in use at Ansonia, Connecticut, I know of only one other copper plant in the United States where gas is used in reverberatory furnaces. This is at the works of the Boston & Montana Consolidated Copper and Silver Mining Company, at Great Falls, Montana, where the gas-fired, tilting-hearth furnaces of the steel-makers have been adapted to the smelting of copper ores. These latter are of the ordinary Butte type, and consist of calcined, pyritous concentrates, and raw, siliceous, first-class ore, the mixture containing some 20 per cent. copper. A 50 per cent. to 55 per cent. matte is produced, and run direct into the ladle of the converters, where it is blown to high blister, and cast direct into anodes for the electrolytic separation of the gold and silver. The slag is poured off by the tilting of the hearth by a hydraulic piston, and is granulated and swept into the rapid current of the Missouri river by a stream of water.

I am not at liberty to publish details regarding the working of these furnaces. From personal observation, I should say that they perform their work satisfactorily and with little hand-labor. I am informed that repairs are light, and that they smelt 45 to 60 tons ore per 24 hours.

The prevailing impression of those metallurgists who have the best opportunities to judge of the work, is, that considering the remarkable results obtained in late years with ordinary, coal-burning reverberatories, the saving in fuel and labor effected by the use of these regenerative, gas-fired, tilting-hearth furnaces is scarcely enough to make up for the greatly increased complication and cost of plant, and the difficulties experienced in obtaining a sufficiently rich gas from the very poor coal (22 per cent. ash) that the producers were expected to employ.

THE "DIRECT METHOD" OF COPPER REFINING.

It has long been known to chemists and metallurgists that when cuprous sulphide and cuprous, or cupric oxide, are fused together in certain proportions, they undergo double decomposition, yielding sulphurous acid gas and metallic copper. In other words, when raw white metal and calcined white metal are melted together, the only solid residue that is left is metallic copper.

These mutual reactions of sulphides and oxides at a high temperature are of the utmost interest to practical men (witness all the reverberatory processes for the smelting of lead ores), and, apart from the great practical and commercial possibilities of this new method, I feel certain that many readers will be glad to have an opportunity of studying the practical application of these reactions. That they have long been known, and much experimented with, heightens, rather than detracts from, the credit of those who have been able to adapt them to profitable ends.

Messrs. T. D. Nicholls and Christopher James, of Swansea, have applied this remarkable reaction to the direct refining of copper matte, without any intervening operation, except the calcination of a portion of the matte, and the process has been in steady operation for some four years in the works of the Cape Copper Company, Ltd., at Briton Ferry, South Wales. Through the courtesy of the directors of this company, and of their assistant superintendent, Mr. Nicholls, I was enabled to see the operation in its regular course, and to study its details.*

The calcination of the white metal is conducted in ordinary revolving cylinders; roasting 10 tons of matte, per 24 hours, down to 2 per cent. sulphur. Various observations on different degrees of calcination led to the adoption of a pitch of calcining from which the calcined metal would take about half its own weight of raw metal to make the refinery charge, as being the most economical; for, although if the metal be calcined to black oxide it will take more raw metal (of course, raw metal, being less expensive than calcined metal), yet the cost of calcining, beyond the point requisite for mixing with half the portion of raw, was found to be more than the advantage gained.

In mixing the charges for the refinery furnace, samples of both calcined and raw metal are taken, and by mixing certain portions of each and melting in a crucible in a Cornish fire, a button of copper is obtained, from the appearance of which it can be seen whether more or less of calcined is needed to give the requisite pitch for furnace work.

Any ordinarily intelligent man will learn how to mix the charges after a trial or two; the resulting button giving very decided evidence as to the proportions being correct or otherwise. An ordinary clay crucible is used, and an iron mold having a half spherical hollow for pouring the button of copper into. If too much raw has been mixed, the button will be solid and flat, covered more or less with regule (sulphide of copper), and if too much calcined has been used, the button will be covered with slag (oxide of copper). The variations from one extreme to the other may be thus described:

Too Much Raw.—Button solid, covered with regule, copper coarse and light in color.

Button solid, surface flat, and blue-black, thin coat of sulphide.

Button vesicular (slightly) and is inclined to climb up the sides

* The following description of the process of "Direct Refining" is largely taken from Nicholl & James' monograph on the subject, and agrees with my own personal observations on the ground. The essential figures are taken from the books of the Cape Copper Company, Ltd., and are vouched for by the secretary and manager. I will not attempt to use quotation marks, as I have altered, condensed, and added to suit present purposes.

of the mold, or to become cup-shaped; on breaking, fracture close, finely granular, and light color.

Proper Pitch.—Button is completely cup-shaped, very vesicular through SO_2 escaping when cooling; it climbs completely up the sides of the mold and has no thick part at the bottom; color very dark reddish-brown on surface, and rough on breaking; the fracture is copper color, just as blister copper.

Too Much Calcined.—Button is copper colored on surface, climbs the mold a little, but has a thick part covered with a blister smooth and dome-shaped all over the button inside the cup edge; button is copper colored, and dome-shaped blister over surface—no cupping of the edges; on breaking, good clean copper color, and pipe-like cavities under the blister. This is too far forward for refinery charge, and any increase of the calcined portion would cover the button with slag.

The trials are simply put in a crucible and melted right away without any cover; when melted, the button is quenched in water. The mixer, after very little experience, can tell from the appearance of the calcined metal, when bruising the sample, pretty nearly how much raw it will take, and one, or at most two, trials are always conclusive. The only precaution necessary is to bruise the samples fine, and thoroughly mix them in order to get reliable results, and 250 to 300 grams is about the weight of a mixed sample for melting.

Upon filling the refining-furnace, the fire is put right and kept going as briskly as possible, the surface of the charge gets melted or glazed, small streams of copper trickle about into any hollows on the surface of the charge, or round the sides of the furnace, and as the heat increases, the copper formed on the surface eats its way down into the charge, setting up the reaction between the oxide and sulphide. The charge (which has now had four or five hours' fire and is getting hot through) swells up and boils gently all over, barm like, or the action may be localized by copper having accumulated in a hollow, and the action starts strongly there and quickly spreads; dense fumes are given off, bursting out through every crack in the furnace; the fire has no draught in consequence of the fumes of sulphur gas completely filling the flue, but the heat generated by the chemical reaction keeps the furnace going until the fumes begin to slacken, the fire is then attended to and heat got up. When the charge is nearly ready for skimming, some slag is removed so as to expose the surface of the copper to the heat, and a strong heat quickly got up. The charge is melted completely off the bottom, and is skimmed, when it is ready for the refiner, exactly the same as if it had been filled with blister or pimple copper; the furnaces being filled every day with the mixed metals, and no difficulty is ever experienced in getting the charge ready for the day men to flap-set it as soon as they come in the morning, and the metal charged takes no more time to flap-set than a blister charge filled into another furnace beside it.

No difficulty is experienced in ladling, the copper is easily refined and keeps its pitch as well as blister, and it takes no more poling than usual; in fact, why should it? But it does take a little more coal, for this reason; a given furnace will burn so much coal and work its charge on it, but a heavier charge of blister can be filled than of metal, because the metal is more bulky than the blister and it contains much less copper, so that less copper results to bear the expense. How far this might be carried has not been determined, but a comparison of 417 charges of metal and 234 charges of blister showed that the blister charges ladled 5.25 per cent. more weight of copper per charge than the metal charges. The men worked piece-work, and the blister was unlimited, while the metal was fixed by the mixing. However, this has been remedied by increasing the size of the furnaces.

The furnaces now melt 15 tons mixed metal every 24 hours; they have done 18 tons for many weeks, but this was found to be heavy for three men to ladle and scarcely enough for four. No difficulty was experienced in introducing the "direct method." The first charge worked as well in every respect as the work done since. It was a complete success; the men falling into the new way with very little trouble; their greatest difficulty lay in the fact that they had been so used to vast quantities of slag being produced from white metal in the roasters, that nothing would convince them but that an equal quantity of slag must come from it when treated in the refinery, and they would pull out something, even if it was half-melted copper. This little hitch was gradually lessened, but all copper men naturally ask, "Where does the slag go to?" forgetting to ask, "How much slag is there in white metal?"

Out of about 20,000 tons of "Best Selected" copper sold during the three years from this process, not a single complaint has been received. It may fairly be said that such a complete and decided improvement has seldom been made with less trouble or expense.

The only additional labor employed being a youth of seventeen, who makes the trial buttons for deciding the proportions of the charges. To show more in detail some comparative results, the complete figures are placed side by side:

1. *"Roasting" twelve months to stock-taking, August 31, 1890 (old method).

2. "Direct" first eight months working to stock-taking, August 31, 1891 (new method).

3. "Direct" seven weeks immediately preceding Dr. Peters's visit to the works, October, 1893 (new method).

FIRST.	SECOND.	THIRD.
	68 tons calcined. 32 tons raw.	74 tons calcined. 26 tons raw.
100 tons white metal 73 per cent., roasted, yielded 51.1 tons ingots. 34.0 tons slags.	100 tons white metal 76 per cent. yielded 68.4 tons ingots. 19.0 tons slags.	100 tons white metal 76 per cent. yielded 71.5 tons ingots. 15.6 tons slags.

This shows that although the calcination has fallen considerably, the process itself, or direct production of copper from metal in the refinery furnace, has slightly improved.

These facts speak well for a complete change in the system of working: that cost no expenditure at all and caused not an hour's delay in the output of the works. It must be at once allowed that to make white metal will cost just the same, no matter what is done with the white metal after it is made. Let it be assumed for comparison that it cost £4 per ton to make white metal, and that to roast white metal and refine the copper costs £2 per ton of copper—these figures are near enough for comparison—then it follows that:

ROASTING.

100 tons white metal cost in making	£400	0	0
And produced by roasting and refining 51 tons ingots, at £2 per ton	102	0	0
Total cost for 51 tons copper ingots	£502	0	0

Or £9 16s. 10d. per ton of copper.

* "Roasting" is the English term for the blister process.

DIRECT.

	£	s.	d.
100 tons white metal cost in making........	£400	0	0
And produced by the direct method 71.5 tons copper at a cost of £1 6s. 3d. per ton, *i.e.*, 13s. 9d. less than roasting.......	93	16	0
Total cost for 71.5 tons copper ingots	£493	16	0
Or £6 18s. 2d. per ton of copper.			

This shows a saving of £2 18s. 8d. per ton of copper in favor of the "direct" method, while at the same time the power of the works for turning out copper was increased in the ratio of 51.1 to 71.5, and that without any expenditure, because it takes the same plant to make 100 tons white metal, whether 51.1 tons ingot copper are produced from it afterward or 71.5 tons ingot copper.

No notice is taken in this comparison of the fact that the roasting gave 34 tons slags to clean, and the direct method now makes but 15.6 tons, less than half. Another fact is worthy of notice. The "direct" process not requiring roasters, no copper is locked up in roaster furnace bottoms, no inconsiderable item in a copper works. Any copper works changing from roasting to "direct," would flow out and turn into cash enough copper to pay for building almost a complete new works, whereas only a few calciners and a crushing plant are required—even where a works has no calciners and crusher already.

For more exact comparisons of cost, every smelter knows his cost of making white metal, and cost of roasting and refining with the copper sold, per ton of white metal made, and it is easy to compare his actual figures with the "direct" results of 71.5 tons copper sold, per 100 tons white metal, and 15.6 tons slags to clean; and, without doubt, the slags can be considerably lessened in quantity.

Without laying too much stress upon the figures obtained at the Briton Ferry Copper Works, where the roasting was undoubtedly not very well done, it must be apparent that calcining part of the white metal must be much cheaper than the much more expensive roasting of the whole of it; while the calcining method enables more of the copper to be produced as ingots, and be at once marketable, instead of being returned through the works again in slags; and the quality of even the high class copper made by the Cape Copper Company has been improved by the change.

This method of producing refined copper would seem to be the one specially suited for treating impure metal, or matte; all volatile impurities would certainly be carried off with the copious discharge of sulphurous anhydride, and as the quantity of slag to be taken off any charge is, by increasing the proportion of calcined metal, almost absolutely under control, the non-volatile impurities will be collected in the slag; especially will this be the case when, as in the direct method, the slag is made all through the charge by an excess of calcined metal, thoroughly mixed, and not only on the surface, as in nearly all other methods of oxidizing impurities in copper.

The calcined metal sometimes gives as much as 3 per cent. of sulphur, and yet it requires just the same quantity of raw metal to reduce it as if it showed very little trace of sulphur. This is probably due to the formation of sulphate of copper, which reacts on the sulphide; sulphate of copper being able to reduce more copper sulphide than the amount of oxide in the sulphate could reduce if mixed as oxide. Commercial sulphate of copper contains 30 per cent. of CuO; 100 tons sulphate should reduce 30 tons of white metal, but we find in the practical smelting of commercial sulphate of copper with white metal that 6 tons of sulphate smelted with 4 tons of white metal gives a coarse copper; probably the reaction may be expressed thus:

$$(CuO, SO_3 + 5\ H_2O) + (Cu_2S) = 3\ Cu + 2\ SO_2 + 5\ H_2O.$$

This would give 6 tons of sulphate as the equivalent of 3.81 tons of white metal; but, in practice, we wanted our copper coarse, and carried more sulphide to insure this coarseness.

Taking an average, it is found that calcination of white metal increases the percentage of the white metal but very slightly, CuO being the same percentage as Cu_2S, and the formation of sulphate (which is no drawback, and therefore not avoided) keeping down the richer produce of Cu_2O.

The best charge we have worked in the refinery gave 12 tons 1 cwt. ingots, and all the slag was wheeled away in one load in a sheet-iron barrow. It was not weighed, but it was certainly not more than 5 cwt. Careful supervision with moderately good copper ore will reduce the slags to a very low point. White metal of 75 to 76 per cent. contains very little slag-forming matter, and, with clean ore, there would be no need to make slag by adding an excess of calcined white metal to wash together the impurities.

Trials with 50 to 55 per cent. metal showed that it was worked very advantageously, a little sand and very little finely ground anthracite coal producing a very nice slag of 6.8 per cent. copper. The copper was good enough as usual.

Moderately clean mattes of 30 per cent. can undoubtedly be worked by this process without bessemerizing, by filling the furnaces repeatedly until there is enough copper to refine, taking care to keep both furnace and charge coarse until actual refining commences.

A study of the foregoing operation and its costs will make it pretty evident that it cannot compete with bessemerizing, where the matte contains separable quantities of the precious metals. For the converter will furnish a 99 per cent. blister in the shape of anodes, ready for the electrolytic department, and the electrolytic copper must be refined anyway, to get it into proper shape. On such matte, the "direct method" would lose its greatest advantage, *i.e.*, the production of refined copper from the matte in one operation.

There is, however, a field possibly open to this method on argentiferous mattes, that seems to me well worth exploiting. Every one who knows anything about the matter is aware that there is a considerable loss of silver in our converter practice, even on mattes carrying only 25 ounces (0.086 per cent.) silver to the ton, or say one-half an ounce of silver to the per cent. of copper. As mattes increase in richness, the losses in silver become more noticeable, though whether proportionately greater we do not know. This loss is apparently small until all the iron is oxidized in the converter, and the charge reaches the condition of high white metal; but from this point on to blister it is very much greater. When lead or zinc are present in considerable amounts, as in copper mattes from the lead cupolas, the silver losses by volatilization are liable to become enormous, and may reach 50 per cent. of the total. It is also probably true that the silver loss, at any stated period of the bessemerizing process, stands in close relation to the blast pressure used in the converter.

In the light of these facts, which seem tolerably well established by our American experience, it seems a question whether the following series of operations might not, in selected cases, be substituted for the present direct bessemerizing process:

1. Bessemerizing our ordinary, argentiferous, 50 per cent. matte up to clean white metal (say 78 per cent.) using trough-converters,

with only about 5 pounds blast, as at the Copper Queen Works at present.

2. Treating this white metal by the "direct process" just described, but so modifying the latter as to avoid tough-poling and ladling, the copper being tapped into anode plates of the highest quality.

From here on, the treatment would be as usual.

By employing this plan, we should probably save two-thirds, or more, of the customary loss of silver in the converter, and we should obtain anode-plates of extreme purity. This latter point would react favorably on the entire electrolytic process, and give a better copper as an end product. It would also save the semi-refining of the blister produced from the converters, which is sometimes necessary, owing to the impurity of the ores.

The principal disadvantages would be the extra cost of the reverberatory operation as compared with simply blowing the white metal (already in the converter) up to blister; the extra plant required; the additional time required, and increased complexity of processes. Local conditions and a more thorough knowledge of our silver losses in the converter will determine the feasibility of this modification.

CHAPTER XVII.

THE BESSEMERIZING OF COPPER MATTES.*

THE first inception, and accurate, practical experiments on the application of Sir Henry Bessemer's process to copper-smelting are due to John Hollway of England.†

Pierre Manhès, of Eguilles, France, can claim the credit for carrying forward the process to a technical and commercial success in the treatment of mattes in France, as the Parrot Company has done in the United States.

His invention, made in 1880, consists mainly in placing the tuyere openings horizontally, and at some distance above the bottom of the converter, so that they may always be above the metallic copper that is formed; or, if blocked, may be easily punched free by an attendant.‡

In the autumn of 1884, the first regular converters in the United States, for the treatment of copper mattes, were erected at Butte, Montana, for the Parrot Silver and Copper Company, by pupils of Manhès.

These gentlemen held that only white metal, carrying from 75

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* This chapter has been written with the collaboration of Mr. H. A. Keller, superintendent of the Parrot Silver and Copper Company, of Butte, Montana. Mr. Keller has run the Parrot smelter, where the copper Bessemer process was first introduced into the United States, during its earlier and partly experimental work, erected the present improved converter plant, and is now engaged in planning extensive improvements in the metallurgical department of the company's works. I am sure all copper men will be glad to join with me in acknowledgments to the Parrot Company for its liberal and enlightened policy in permitting the publication of so much of their dearly-bought experience and results. Mr. Keller has furnished most of the more practical and valuable material in this chapter.

† John Hollway, *Treatment of Sulphides*, British patent No. 4,549, November 9, 1878.

‡ "Bessemerizing Copper Matte," by Prof. T. Egleston, *School of Mines Quarterly*, New York, May, 1885. *Bessemern und Electrolyse*, by C. A. Hering, Freiberg, 1886, p. 16.

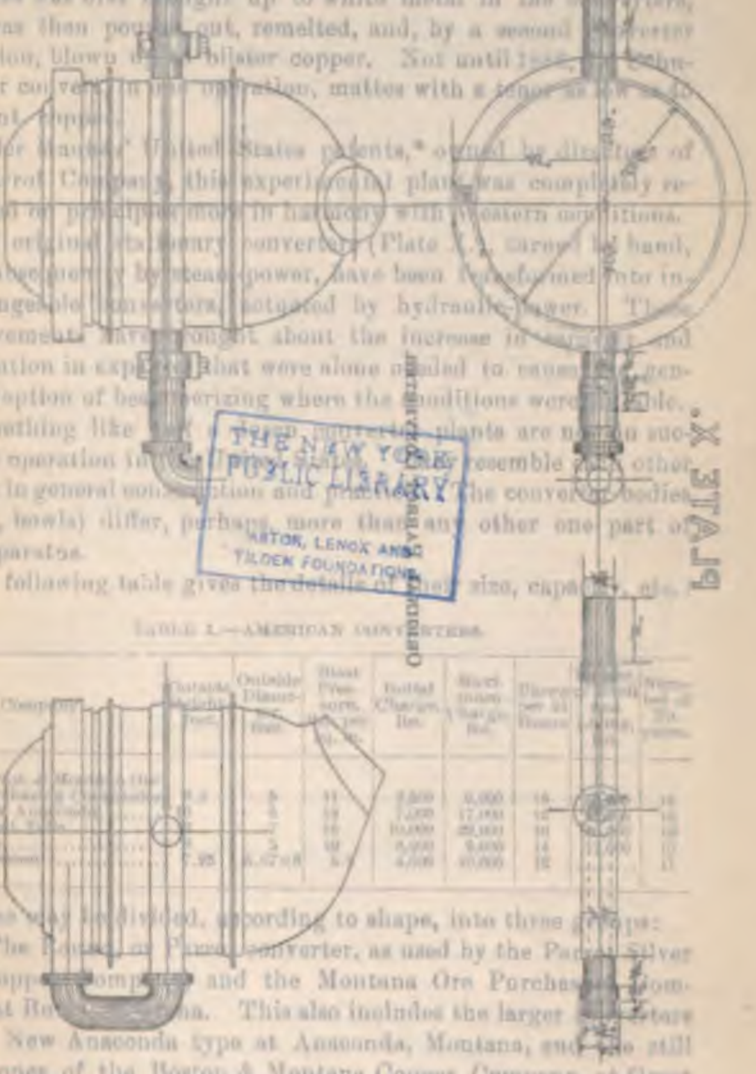
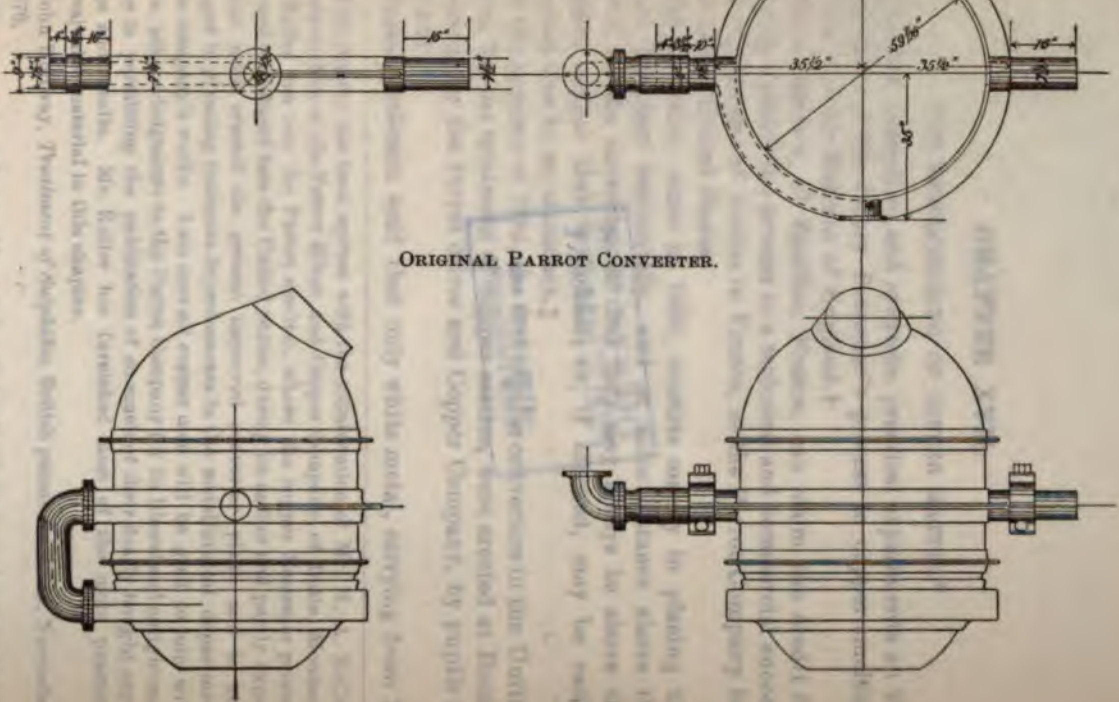


PLATE X.

PLATE X.



ORIGINAL PARROT CONVERTER.

per cent. to 80 per cent. copper, could be blown up to the required metallic condition as blister copper in one operation. In consequence, the 40 per cent. to 50 per cent. matte from the smelting-furnaces was first brought up to white metal in the converters, and was then poured out, remelted, and, by a second converter operation, blown up to blister copper. Not until 1886, did Schumacher convert, in one operation, mattes with a tenor as low as 45 per cent. copper.

Under Manhès' United States patents,* owned by directors of the Parrot Company, this experimental plant was completely remodeled on principles more in harmony with Western conditions.

The original stationary converters (Plate X.), turned by hand, and subsequently by steam-power, have been transformed into interchangeable converters, actuated by hydraulic-power. These improvements have brought about the increase in capacity and diminution in expenses that were alone needed to cause the general adoption of bessemerizing where the conditions were suitable.

Something like half a dozen converter plants are now in successful operation in the United States. They resemble each other closely in general construction and practice. The converter-bodies (shells, bowls) differ, perhaps, more than any other one part of the apparatus.

The following table gives the details of their size, capacity, etc.:

TABLE I.—AMERICAN CONVERTERS.

Company.	Outside Height, feet.	Outside Diameter, feet.	Blast Pressure, lbs. per sq. in.	Initial Charge, lbs.	Maximum Charge, lbs.	Blows per 24 Hours	Weight of Shell and Lining, lbs.	Number of Tuyeres.
Parrot Type. Parrot, & Montana Ore Purchasing Companies	8.5	5	11	2,500	9,000	16	16,000	16
Parrot Type. New Anaconda........	10	6	18	7,000	17,000	12	22,000	16
Parrot Type. Great Falls............	13	7	16	10,000	22,000	10	26,000	18
Stalmann..................	8	5	10	8,000	9,000	14	17,000	10
Copper Queen...............	7.25	5.67×8	5.5	4,000	10,000	12	....	11

These may be divided, according to shape, into three groups:

1. The Round, or Parrot converter, as used by the Parrot Silver and Copper Company, and the Montana Ore Purchasing Company, at Butte, Montana. This also includes the larger converters of the New Anaconda type at Anaconda, Montana, and the still larger ones of the Boston & Montana Copper Company, at Great Falls, Montana.

* Nos. 470,384 and 470,644, both dated March 8, 1892.

2. The Square Stalmann, or Old Anaconda converter (nowhere in use at present).

3. The Cylindrical, Trough, or Modified Leghorn pattern, introduced lately by Douglas at the Copper Queen Mine, Bisbee, Arizona, and, in an original form, by Raht at the Philadelphia Smelting and Refining Company's works, Pueblo, Colorado.

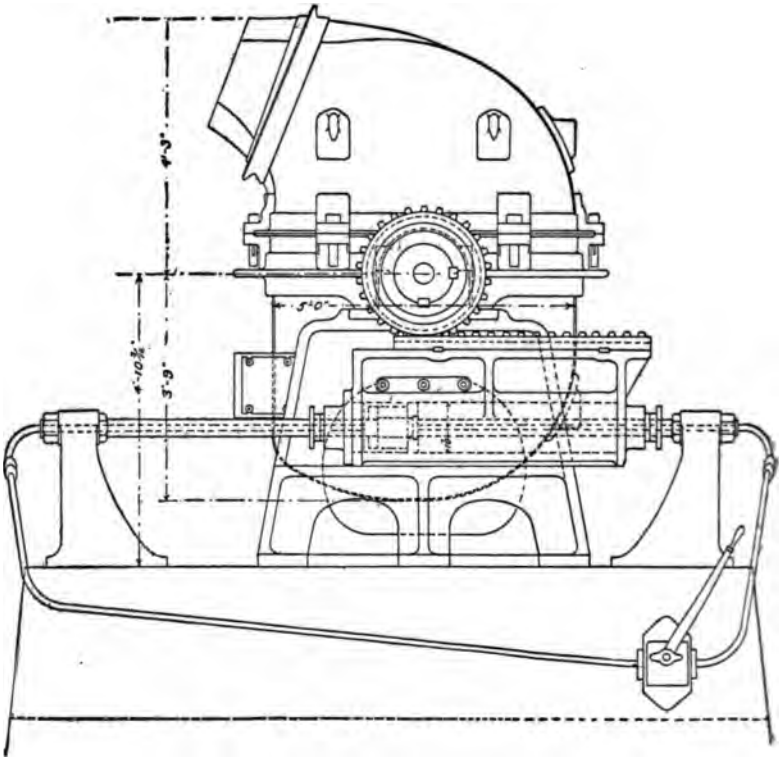


FIG. 70.—THE STALMANN CONVERTER.

As our description of converter work will refer mainly to the ordinary round, or Parrot type, of converter, it will be most convenient to first refer briefly to the Stalmann and the Copper Queen converters.

The peculiarities of the Stalmann converter are well shown in the accompanying illustration. Its square form and peculiar shape are intended to lessen the corrosion of the lining and the blowing out of molten material. After a very extensive experience with it, the Anaconda have returned to the round type of converter.

The object of the trough, or Copper Queen, converter is, to admit of a gradual and steady tilting of the bowl during the opera-

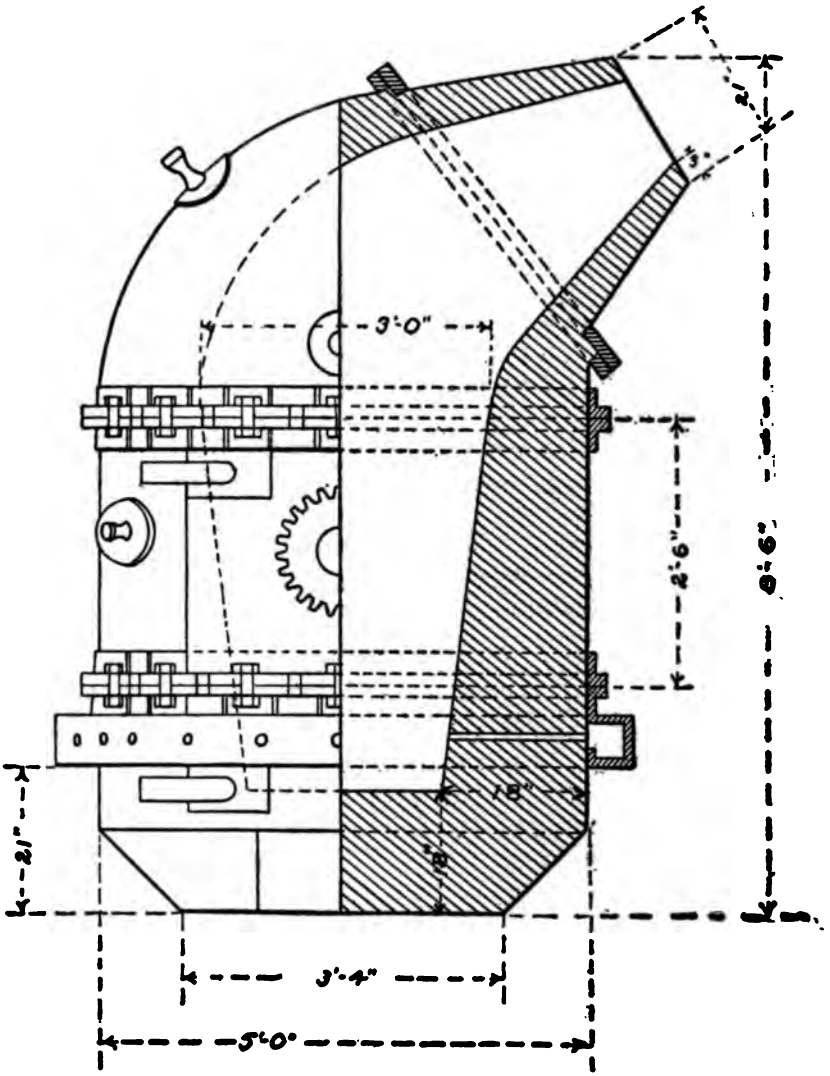


Fig. 71.—The Parrot Converter.

tion of blowing, so that the blast may be forced only through the superincumbent layer of matte (the reduced copper settling at once to the bottom of the vessel), thus requiring less pressure and

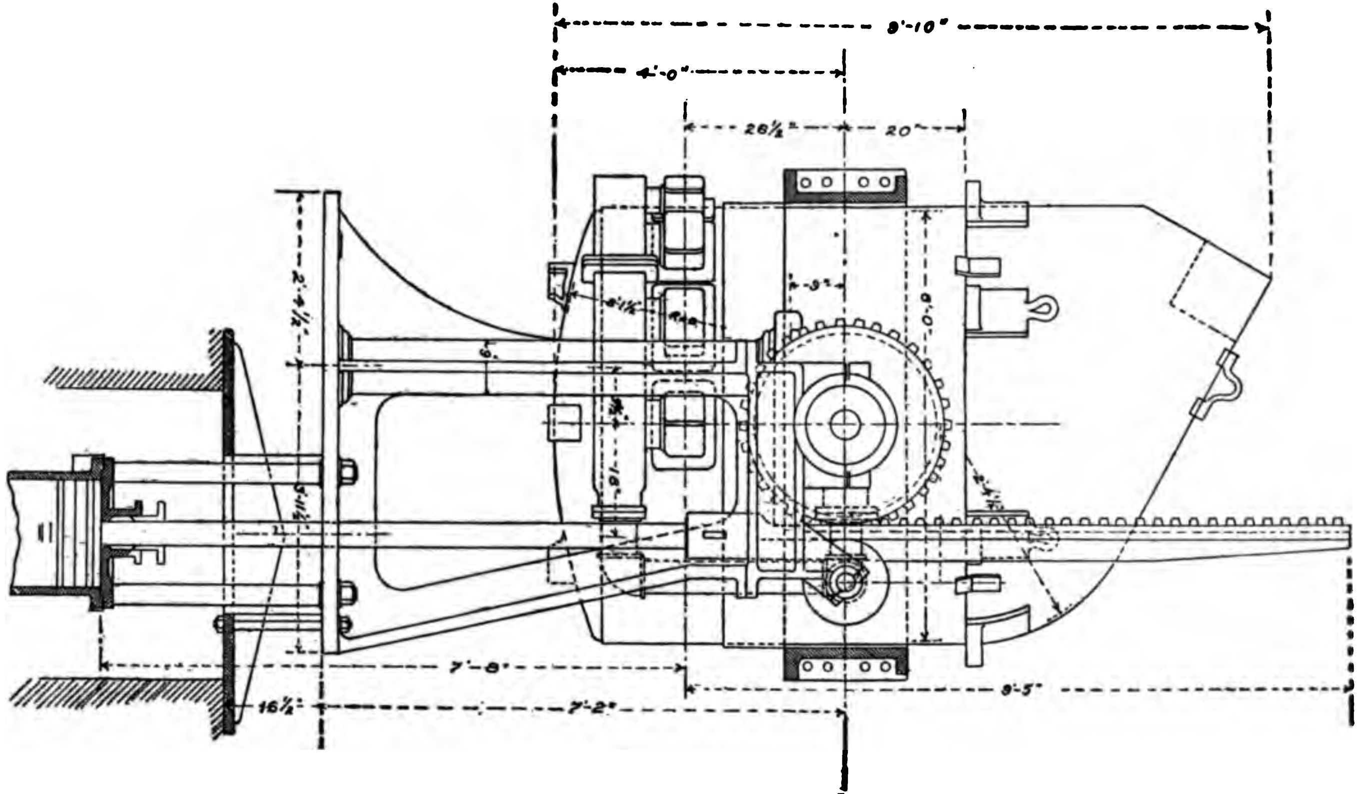


FIG. 72.—ANACONDA CONVERTER.

avoiding the action of the blast on material that requires no further oxidation. Theoretically, it would seem that a thick layer of matte with high pressure of blast would be the more economical plan, as the current of air is thus kept for a longer time in contact with the substance to be oxidized. But the actual daily results,

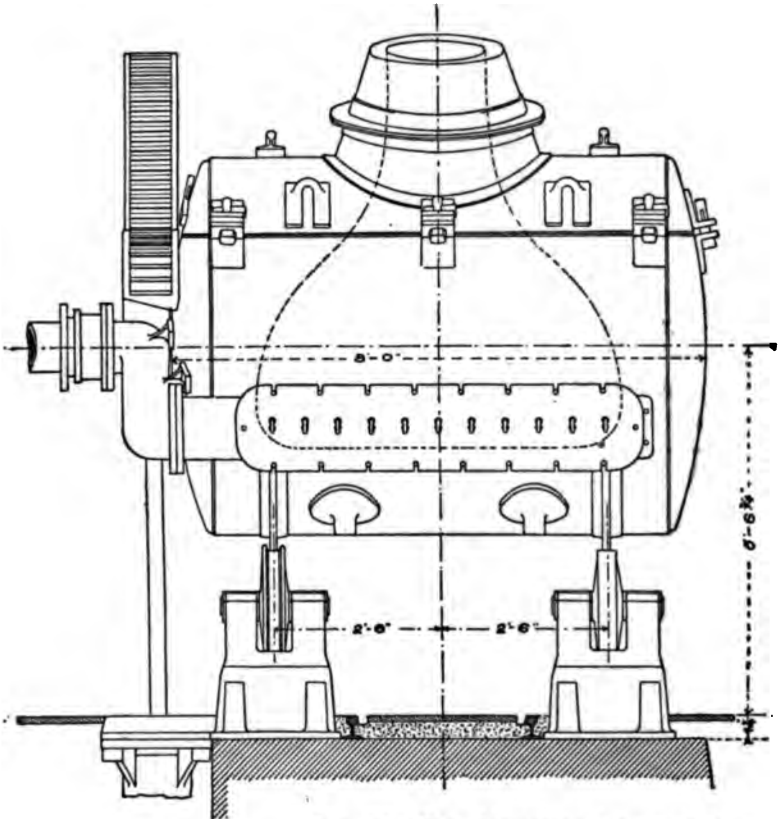


Fig. 73.—The Modified Leghorn Converter.

as obtained by this converter at the Copper Queen mine, and given me by Mr. Douglas, seem to indicate that a light blast and small volume of air will do far more work than has generally been supposed. It is stated that, with an average blast pressure of 5½ pounds per square inch, a single converter is turning out 40,000 pounds of 98.5 per cent. blister copper per 24 hours, from matte averaging 51 per cent. copper. The average charge of matte is about 7,000 pounds, and this is blown to blister copper in 90 minutes. If these results are confirmed by longer experience, it is quite possible that the costly air-compressing engines of the present converter plants may be replaced by the inexpensive Root, or Baker, blowers, thus effecting a decided saving in the first cost of

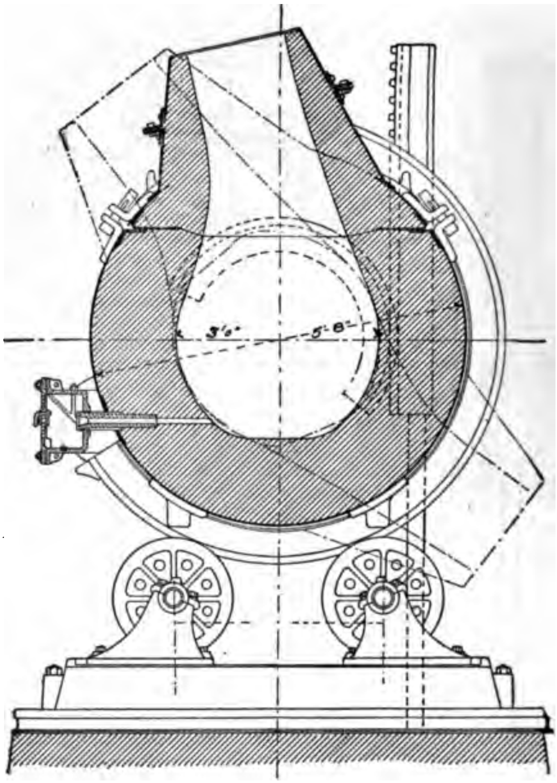


Fig. 74.—The Modified Leghorn Converter.

a converter plant and reducing the blast-bill from some 20 per cent. to perhaps 7 per cent. of the total expense of converting.

PECULIARITIES OF COPPER BESSEMERIZING.

There is a wide difference between the Bessemer process as applied to iron, and the same operation when employed for the concentration of copper mattes.

In the former case the operation is comparatively simple. The molten iron is a homogeneous product; and what is more to the

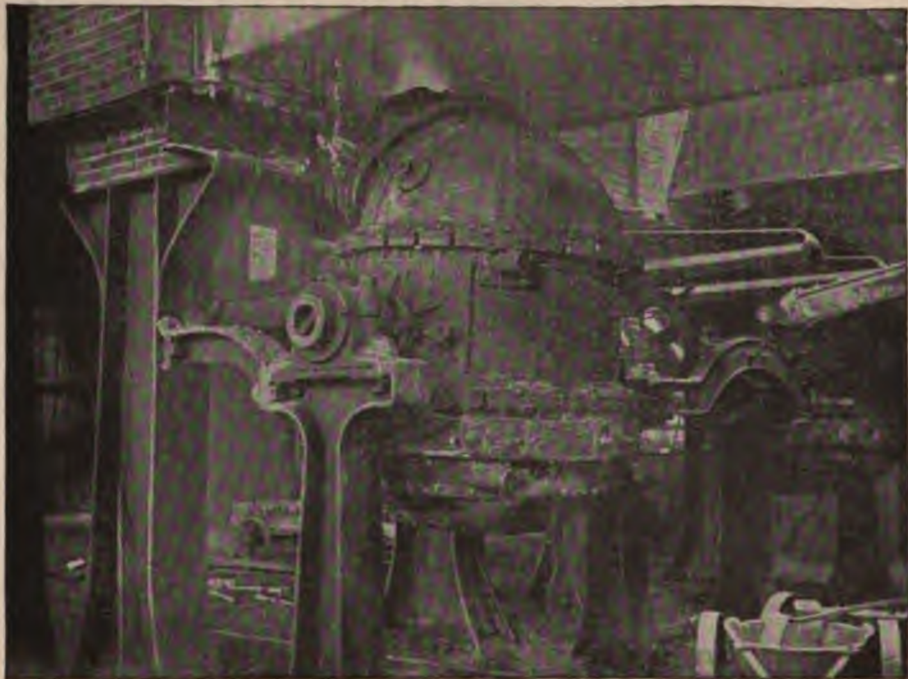


FIG. 76.

purpose, remains in this homogeneous condition during the entire process.

There are only 3 per cent. to 6 per cent. of foreign substances to be oxidized. And these small quantities of silicon, carbon, etc., furnish ample heat for the entire operation, while they are present in such trifling amounts that the mass of metal scarcely shrinks perceptibly after their removal.

As the carbon is burned to a gas, and the blowing is stopped before much if any iron is oxidized, there is but a trifling quantity of slag formed, while the lining of the converter is very slightly corroded.

In bessemerizing ordinary copper mattes of 45 per cent. to 55 per cent., the oxidation of the iron forms large quantities of basic slags, which at once attack the converter-lining to satisfy their strong affinity for silica.

As soon as the operation has proceeded so far that there is not sufficient sulphur left to hold all the copper in combination, we begin to have three distinct products in the converter, all of different weights, chemical qualities, and specific heat. The slag floats on the surface; in the middle will be found a constantly decreasing zone of matte; and, lowest of all, a constantly increasing layer of metallic copper.

Manhès' plan of using horizontal tuyeres, and raising them above the bottom of the converter, so as to leave a chamber for the quiet subsidence of the metallic copper, remedied certain of these difficulties; while the pouring-off of the accumulated slag at about the middle of the operation prevents the troubles that arise when the metal is heavily blanketed with this material.

The two weakest points in modern copper-bessemerizing are:

1. The rapid destruction of the lining, which is eaten out within 12 blows, or less, by the ferrous oxide resulting from the oxidation of the matte.

2. The large amount of slag that has to be retreated; nearly all the converter slag containing too much copper to throw away. Where it is not needed as a basic flux, it adds a considerable item to the cost of treatment.

The exact extent of these disadvantages, as well as the best means to lessen them, will be discussed when we come to the practical running of converters.

MATERIALS SUITABLE FOR BESSEMERIZING.

Hollway's experiments were conducted largely upon heavy, pyritic ores, and though he attempted the concentration of the resulting matte, his results show the futility of trying to go too far in one operation.

The fusion of *ores* by the combustion of their own oxidizable constituents will be found treated at length under "Pyritic Smelting."

In American practice, only *mattes* are subjected to the operation of bessemerizing; and it is important to determine to what degree of concentration it is most profitable to bring the matte, before it is sent to the converters.

This is a somewhat complicated question, involving a careful balancing of the advantages and disadvantages of making:

(*a*) A comparatively low-grade matte, and doing a large amount of concentration in the converters; or,

(*b*) A high-grade matte; thus giving the converters but little work to do, while the main labor is thrown upon the calcining furnaces.*

The mere fusion of the ore, either for a high, or low-grade matte, is a comparatively simple and straightforward operation. But a rare combination of both metallurgical and commercial knowledge is required to produce matte of the most profitable degree of concentration, when all stages of the process are considered.

In other words, one of the most important questions that a metallurgist is called upon to answer to-day, when treating sulphide ores by fusion, is:

Is it more profitable for me to burn off the sulphur contained in my ores while they are in a solid, or molten condition?

Each case has to be decided upon its own merits; the character of the ore and combustibles, the cost of fuel and refractory materials, and the existence of water-power, being the main factors that bear upon it.

About the heaviest expenses of bessemerizing are the power required for the blast, and the refractory materials for the linings.

Hence, with ample water-power, suitable quartz and clay, and expensive fuel, one would naturally incline toward the production

* Types of these two systems of working may be seen just now at the Boston & Montana's smelter at Great Falls, and at the Montana Ore Purchasing Company's smelter at Butte.

of a comparatively low-grade matte. (A matte of 45 per cent. to 50 per cent. is considered low-grade, when referring to the Bessemer process as usually carried out in America.)

Under the above conditions, a large converter-plant would be necessary and economical. This means, however, the production of a very large amount of rich converter-slag for retreatment, which may increase the expenses heavily. On the other hand, this slag will usually be a welcome flux for the raw, or only partially roasted, calcines. For most ores in this condition are too siliceous to smelt alone. This is the case even when they consist largely of pyrites-concentrates, carrying perhaps 38 per cent. iron. For this iron, being in combination with sulphur, is not available for slag-formation, and simply goes into the matte as a sulphide.

Hence, the value of a heavy, ferruginous converter-slag as a flux, and when the cupola-charge consists largely of ores or concentrates in a finely-divided condition, this slag is so beneficial mechanically, as well as chemically, that its addition may actually reduce the total cost of smelting the ore, instead of increasing it.

With cheaper fuel and no available water-power, it is generally better, especially where fine ores or concentrates are concerned, to throw the burden of the work upon the calciners, thus producing a high-grade matte at the first smelting, say 55 per cent. or thereabouts.

The present automatic calcining furnaces run at an extremely low cost per ton of ore, and reverberatory furnaces have been so improved, as regards capacity and economy, that we need not hesitate to employ them for pulverized calcines, even in direct competition with the blast furnace.

With a plant of this description, care must be taken not to remove too much of the sulphur, or the resulting matte will be so high-grade as to enrich the slag beyond the point of economy.

This highly-concentrated matte can be treated in a small and inexpensive converter-plant, and, as it contains but little iron, will greatly reduce the cost of linings.

For the same reason, it will produce but little slag for retreatment—a favorable circumstance—as the strongly calcined, pyritous ores will require no basic flux, their contents in iron being mostly oxidized, and thus in a condition to combine with silica at once.

It is sometimes difficult to produce matte of a sufficiently high grade for bessemerizing, in a single smelting. This is the case with a large class of pyritic ores, high in sulphur and low in copper.

After being subjected to the most careful heap, or stall-roasting, they still contain a little too much sulphur to yield the necessary 45 per cent. to 50 per cent. matte for bessemerizing. The ordinary matte will run 30 per cent. to 37 per cent., we will assume, which scarcely seems low enough to warrant the sandwiching-in of a costly and tedious operation of calcination and concentration, to fit it for the converters.

We have, it seems to me, three methods at our disposal, which may bridge this little gap between mattes unfit, and fit, for converting.

(*a*) Assuming that we are employing blast-furnace smelting, preceded by heap or stall-roasting. One of the main influences in producing a low-grade matte in this operation is the large amount of unroasted fines that accumulate in excess of the capacity of the heaps to utilize them, as well as from the covering of the heaps or stalls.

By erecting some of the modern automatic furnaces, we could calcine these fines almost dead, about as cheaply as we roast the coarse ore, when time and loss is considered. They would then act powerfully in enriching the matte, instead of diluting it, as before; and in many cases would at once raise it to the point for profitable bessemerizing.

(*b*) This suggestion refers to a case similar to the one just imagined, but proposes to effect a rapid and economical concentration of the low-grade matte by means of the operation usually known in America as "Pyritic Smelting." Matte is an ideal substance for this process, and can be rapidly concentrated from 30 per cent. up to 50 per cent., or 60 per cent., in a blast-furnace, without the addition of any fuel, except that necessary to heat the blast. (See "Pyritic Smelting.")

(*c*) We may remove the excess of sulphur by running the blast-furnace as a partial oxidizer. This involves no hot blast and no radical change of practice, and is a plan that should be carefully considered by metallurgists who are confronted by the difficulty alluded to.

At the present time, however, in America, no copper material is regularly bessemerized excepting mattes containing over 45 per cent. copper. These are commonly blown up to blister copper in a single operation, interrupted only by a single skimming, or pouring of the slag. Lower-grade mattes corrode the lining too rapidly; while mattes over 60 per cent. do not furnish sufficient heat by their oxidation to comfortably complete the process.*

DESCRIPTION OF A CONVERTER PLANT.

In describing a typical, modern converter plant, special reference is made to the last plant erected at the Parrot smelter by Keller, and which is shown in Plates XI. and XII. Details and dimensions are better seen in Plates XIII., XIV., and XV., which show a two-converter plant constructed subsequently by the Parrot Company, at their Western Iron Works, for the Montana Ore Purchasing Company.

A glance at these drawings shows that such plants are essentially composed of two distinct parts, namely: the remelting, and the converting departments. Though separated, it is of the utmost importance that these should be able to operate in unison; for delays occurring in either materially affect the good working of both.

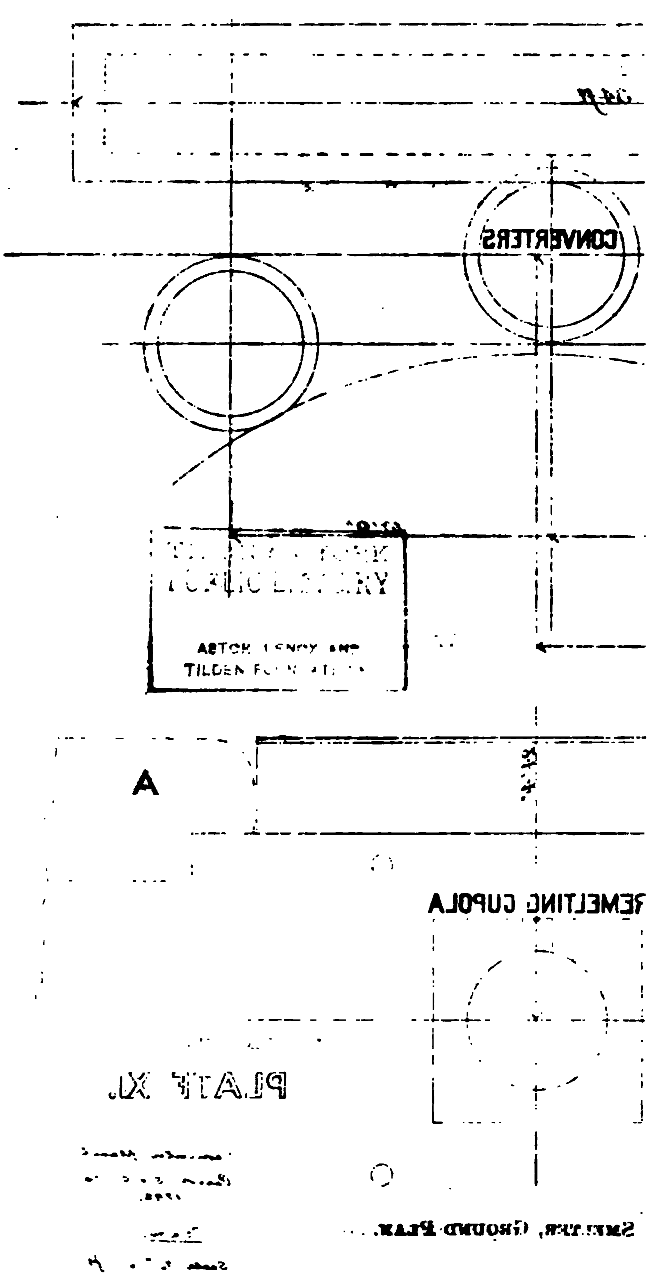
Such a plant is best placed in a well-ventilated iron building, with the highest point above the cupolas, thus making the two roof-slopes of very unequal length.

The principal aim in planning such a battery of converters is to provide for their continuous, free running, by keeping the remelting cupola in steady operation; for intermittent working is likely to block the latter and thus delay the converters. This is one of the most difficult matters in managing converters.†

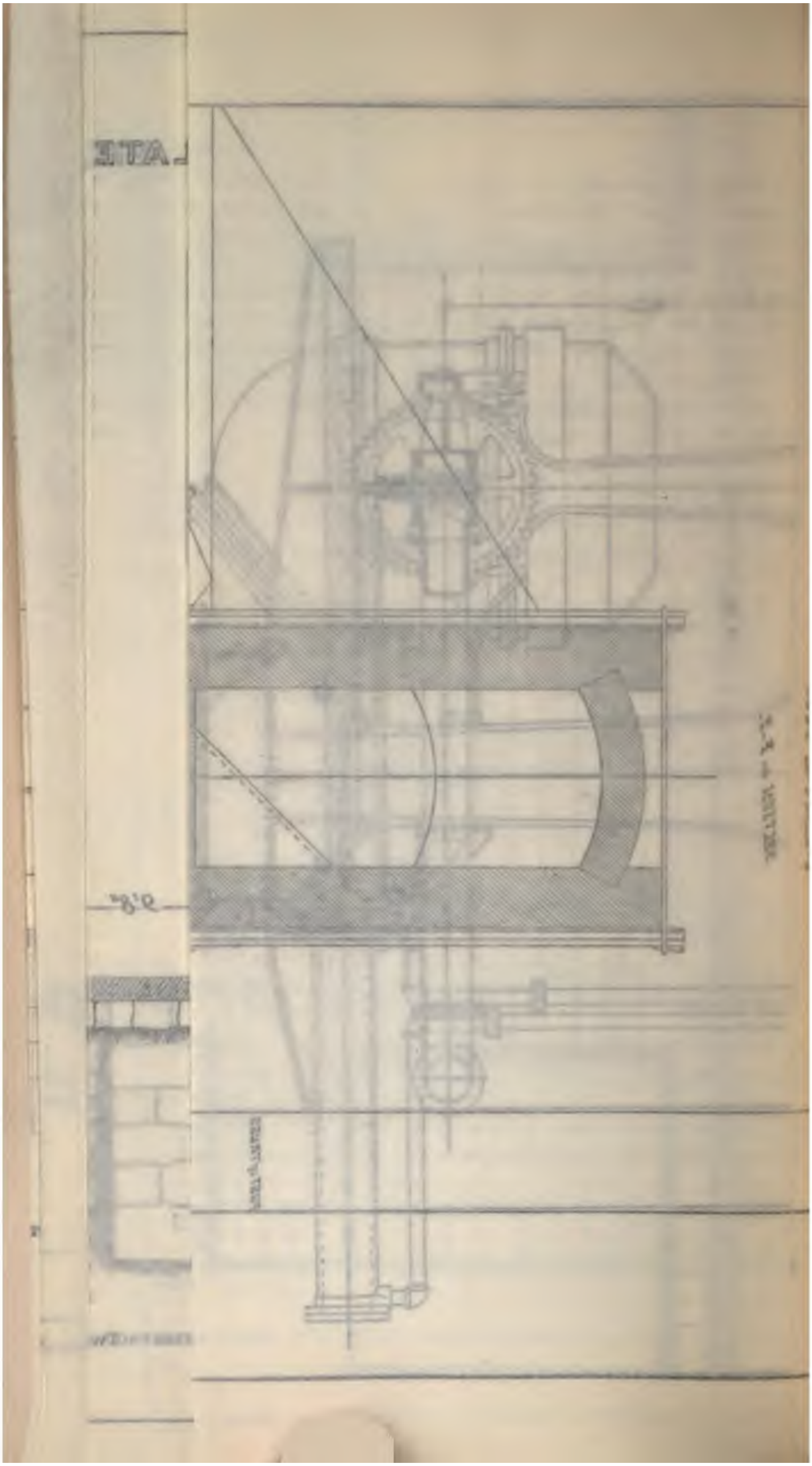
The entire proposition is much simplified by selecting a location that will permit the complete discarding of all remelting appliances. This has been done at the Boston & Montana smelter, at Great Falls, Montana, these having been constructed at a time when bessemerizing had advanced sufficiently far to permit the adoption of such a plan without any fear of failure. There, the large converters are placed directly in front of the gas-fired reverberatory furnaces, the latter serving as storage receptacles for the molten matter to be bessemerized. Longitudinally, between the

* The heat units freed by oxidation are, for sulphur, about 2,200, and for iron, about 1,576. To these should be added the heat resulting from the combination of the iron and silica, while a considerable deduction must be made for the disassociation of the sulphides.

† All this trouble could be obviated by the employment of a reverberatory forehearth, as explained in the section on "Forehearths."



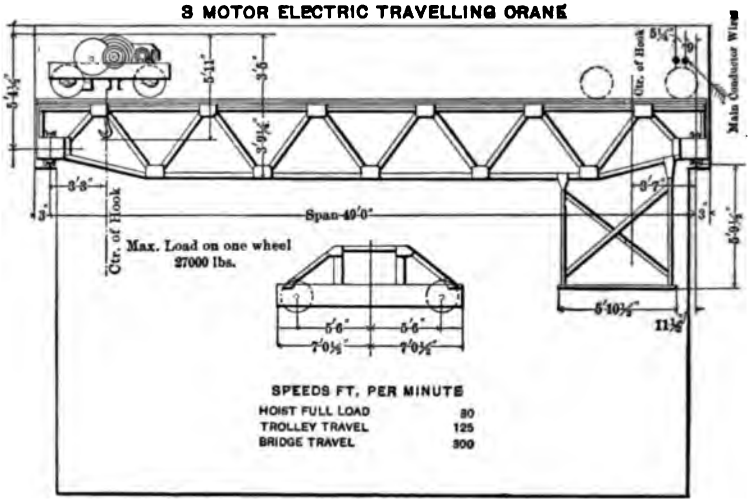
CONVERTERS
A
REMELTING CUPOLA
PLATE XI.
SMELTER, GROUND PLAN.



reverberatories and converters, runs a large, electric traveling-crane, with auxiliary quick lift, shown in the accompanying cut. (The quick motion is used for tilting the ladle and other light work.) This crane also brings the matte from the ore blast-furnaces, where it is collected in their large forehearths, described fully in the section on "Forehearths." It should be in duplicate, and is used for handling all materials and also for lifting the converter-bodies.

The remelting of the matte for the converters is done in a small, circular cupola-furnace provided with a good sized dust-chamber.

3 MOTOR ELECTRIC TRAVELLING CRANE



In the plant under discussion, it is connected with a stack 4½ feet square and 100 feet high, which also receives the fumes from the three converters.

This furnace is situated directly in front of the converters and sufficiently above the latter to permit the molten matte to run into them by its own gravity. It is hung from a cast-iron frame, and has its charging-floor arranged and supported in the customary manner. The upper portion of its shaft consists of a sheet-iron ring lined with fire-brick, below which is placed a circular water-jacket. The blast for two such smelting furnaces, together with some little wind required for drying out converter linings, is supplied by a No. 6 Root blower, making 100 revolutions per minute. The crucible of this furnace is completely independent, and serves merely as a storage-vessel for accumulating the molten matte for the next converter charge. It consists of a sheet-iron ring, forming a truncated cone, and rests with its narrower end upon a circular, cast-iron base-plate, mounted on trucks. It is thus easily replaceable. This sheet-iron ring is lined thickly with a tamping mixture of 20 per cent. plastic clay and 80 per cent. of the same crushed quartz that is used for lining the converters. (All these percentages of lining mixtures refer to volume, and not to weight.) The base-plate requires but a thin coating of this gannister, as it chills sufficiently by radiation. The crucible ring is made strongly flaring toward the top, so as to allow the lining to be readily patched at the point where it is mostly exposed to the oxidizing action of the four tuyeres.

With converters the size of those under consideration, the height of the cupola floor above the converter floor is 7½ feet, the tap-hole in the cupola-crucible being two feet more. The total height of the remelting cupola is about 13 feet, of which, the mounted well, or crucible, occupies 5 feet. The latter delivers its liquid contents to either of the three converters through a clay-lined, sheet-iron launder, or spout. To ensure a quick and lively flow of the matte, the fall should not be less than 2½ feet in 20 (12½ per cent.), and rather more than less. This spout should be hung to move freely, as well as to admit of rapid vertical adjustment; which latter is accomplished either by a pillow-block, or by a turn-buckle with high-pitched (steep) thread.

The converters are arranged segmentally, so that the one spout may serve for all three, see Plate XI. By providing one cupola for each set of three converters, it is the intention to always keep two of the latter in blast. In order to lose no time between charges, the third converter is held in readiness to be charged when one of the other two is turned down for casting the copper. Indeed, if the cupola gets ahead of the rest of the plant, all three converters are, at times, run simultaneously, which is found to be much better practice than to check the cupola by partially shutting off its blast.

Each converter-mouth blows its gases into an opening, or hood, which leads through a canal, or pipe, to the main dust-chamber. Since these openings unavoidably admit much cold air, the stack must not only be high, but each canal must be kept isolated until it reaches the common dust-chamber. Otherwise, suffocating fumes will issue from the third opening, which is temporarily not in use. These flue-openings should be provided with heavy, steeply-inclined iron plates, which admit of being easily cleansed from the blown-out material that lodges and accumulates rapidly upon them.

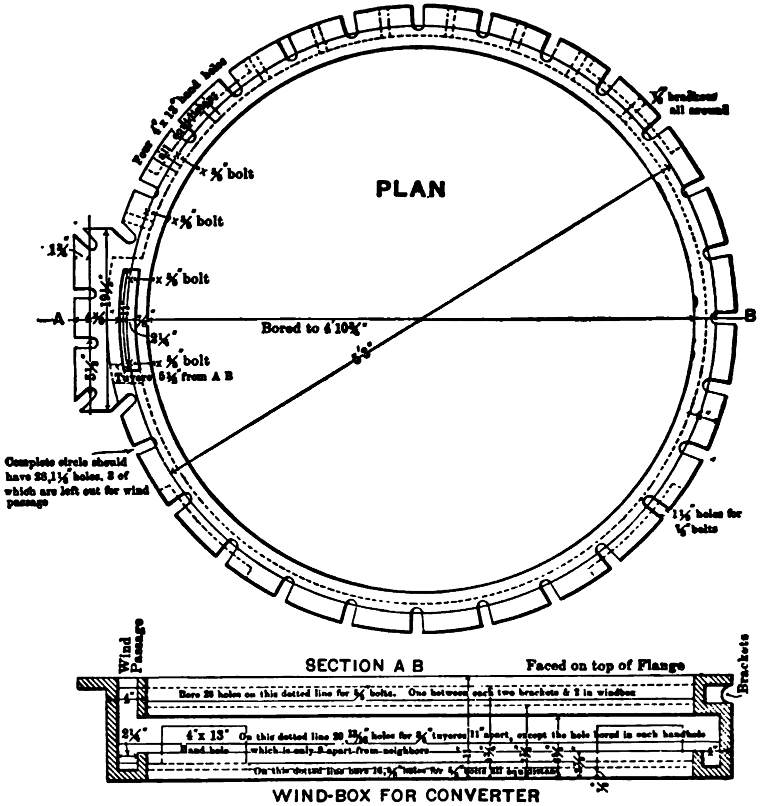
A battery of converters, therefore, consists of three stationary stands, each capable of accommodating one replaceable body. For each stand, two bodies, or shells, should be provided, besides an extra shell at large, making seven in all. In addition, there should be two extra tops. This provides amply for keeping shells lined in advance, as well as for possible repairs on the other shells.

The stands are composed of two cast-iron columns, braced (at the Parrot smelter) against still larger columns that support the overhead brick fume-canals leading to the main dust-chamber. The summit of each stand-column carries a bearing for the purpose of accommodating the trunnions to which the body is bolted. One of these is cast solid, but shifts in the bearing to fit the converter-lug, while the stationary one is hollow and conducts the blast to the wind-chest, a stuffing-box being introduced to make an air-tight connection.

The cast-iron ring (see illustration), forming the wind-box, contains 16 horizontal drilled holes, $\frac{13}{16}$ inch in diameter. These tuyere holes are closed on the outside with a wooden plug during the course of the operation. This is removed to allow the cleaning or "punching" of the openings with an $\frac{11}{16}$-inch iron rod, to free the tuyeres from chilled matte or slag. Through carelessness on the part of the workmen, the front tuyeres (which, of course, come lowest when the vessel is turned down) are sometimes not only plugged themselves, but have the corresponding portion of the wind-box filled with matte or metal. To avoid this, several of these front tuyere-holes are sometimes omitted, a proceeding that scarcely seems necessary; for, as long as they are unfilled, they assist in distributing the blast.

The burning-out in, or near, the wind-box, directly above the tuyeres, is the greatest source of damage to shells. The ring is provided with openings for the clearing out of any material that may block it. While running, these openings are covered with sheet-iron plates, bolted through the ring. To lessen still further the danger of filling the wind-chest, the tuyeres at Anaconda have been placed entirely outside of the ring; at first, below; latterly, above it. In the latter case, we prevent the ring from hiding the very spot where the lining suffers mostly from corrosion. It has, also, the further advantage that the connection leading from ring to tuyere-box may be provided with a ball-socket opened automatically by the blast pressure, and closed by ball as soon as the tuyere stops from any cause.

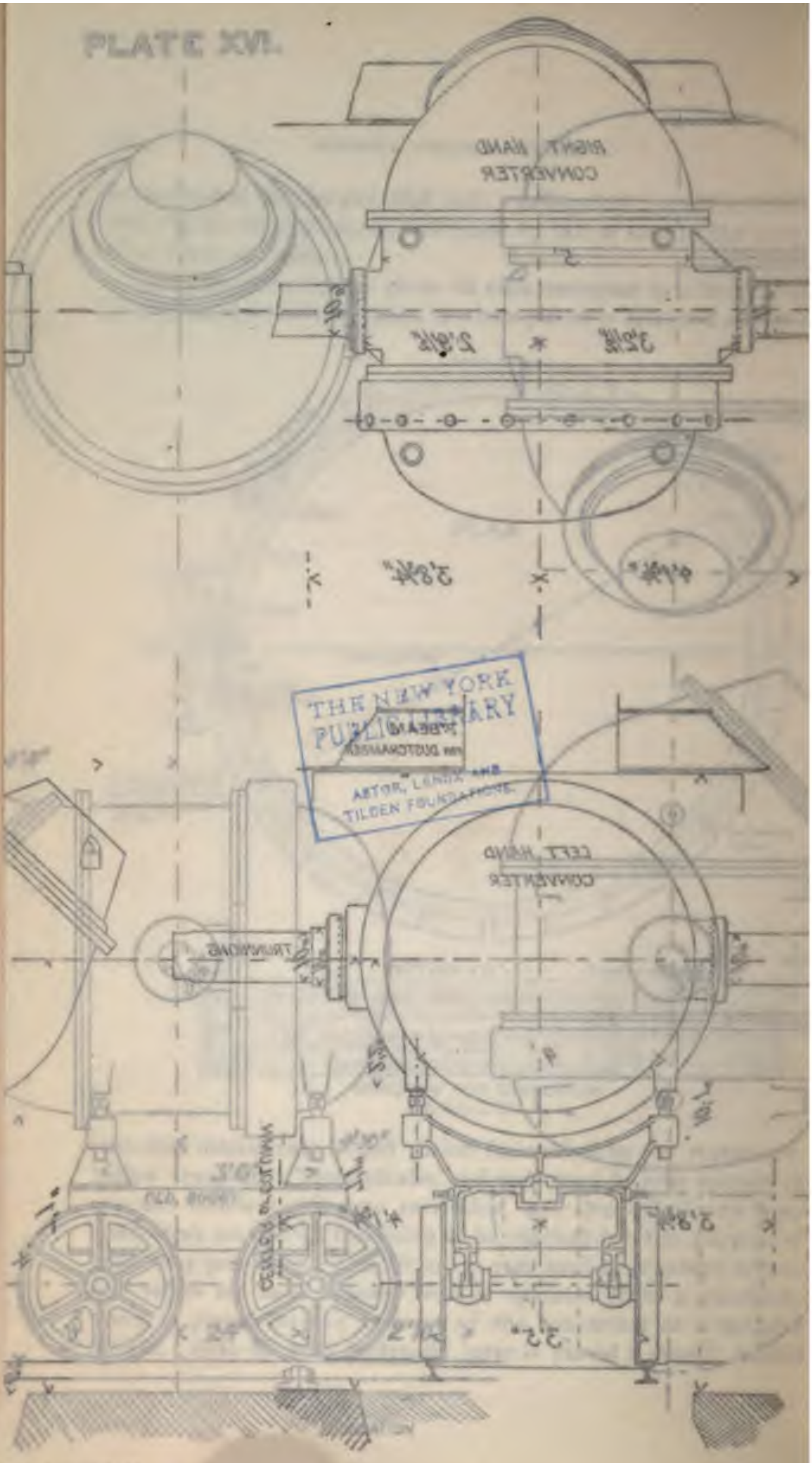
The tilting motion is given to each converter by a horizontal, or vertical, hydraulic cylinder, to whose piston is attached a toothed



WIND-BOX FOR CONVERTER
P. S.& C. CO.

rack that meshes into a steel pinion fastened upon the stationary, hollow trunnion. The cylinder and rack must be long enough to give the bodies a complete revolution, and thus allow any especially weak portion of the lining to be reached for the purpose of temporary patching. The tilting is regulated by separate critchlow valves for each cylinder, and is operated from a platform, which is placed directly in front of the converters at a suitable height. Side of each hydraulic lever is placed a second handle

PLATE XVI.



for opening and closing the air-cock that controls the blast to the converter.

The hydraulic pressure is furnished by a small plunger pump, requiring very little power. The required water pressure depends, of course, upon the size of the tilting-cylinders, which, at the Parrot, make a sufficiently quick motion with a pressure of only 180 pounds to the square inch.

The exchanging of converter-bodies is done by loading them upon buggies specially constructed for horns, or lugs, with which the converters are provided, and which are illustrated in Plate XVI.* The bed of this buggy has a ball-joint admitting of some lateral motion, while the vertical adjustment is effected by heavy, threaded bolts. The car runs on rails to suitably located turntables, where it is delivered to jib-cranes for further handling.

With a traveling crane, the buggies are entirely done away with, the converter-bodies being simply lifted in and out, and secured to each trunnion by a single wedge instead of requiring six bolts.

The floor of the converter-house in the vicinity of the stand should be made of heavy cast-iron plates, while an inexpensive block pavement of wood will be found the most durable about the cranes, where heavy lifting takes place. At the Parrot, this is made from the waste ends of the mine timbers, cut six inches long. A few chips are thrown upon the wooden pavement where a hot converter is to be set down.

The converter-bodies are usually made up of three separate parts held together by bolts, viz.: the top or helmet (with detachable snout), the center, and the bottom. To the latter, a cast-iron ring is attached, forming the wind-chest. All portions of the shell are now universally made of strong boiler iron, with cast-iron trimmings. The thickness of the boiler iron varies with the size of the converter. Those under consideration are $\frac{5}{16}$ inch thick for the two lower sections, and $\frac{1}{4}$-inch for the top. The weight of the complete (unlined) shell is 6,000 pounds.

Each two sections are held together with planed, cast-iron flanges, provided with a sufficient number of $1\frac{1}{4}$-inch bolts to furnish enough strength to hold them together even if several bolts have to be omitted, owing to injury to the flanges. The double flange securing the top to the center section may do

* Fig. 75 shows a hydraulic transfer-car, designed and manufactured for this purpose by Fraser & Chalmers.

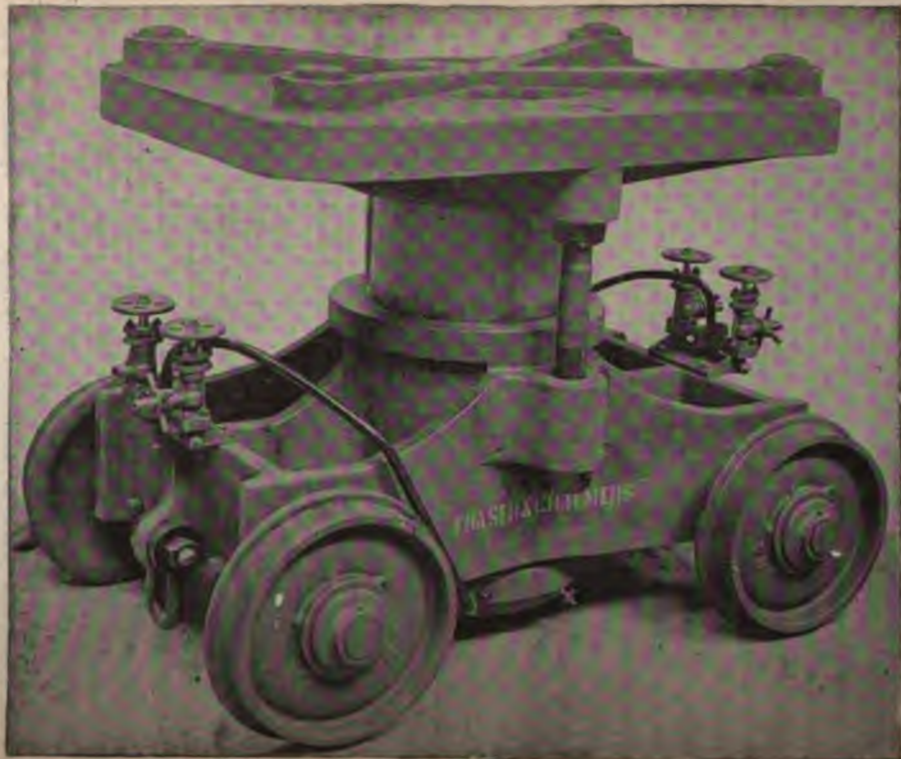


Fig. 75.

with 14 bolts, but a somewhat larger number should be used for the lower junction to ensure a tight joint near the wind-box. Since the lower two sections are rarely taken apart, they have sometimes been made into one. This construction can hardly be recommended, since most wear is in the bottom section near the wind-chest. Hence, separation may greatly facilitate repairing, besides demanding smaller patches.

The tendency of late has been toward the use of larger converters, which means, of course, an increased scale of operations all the way through. This increase of size is, however, limited by the following considerations:

1. Very large converters require very powerful cranes for handling them. These, in turn, necessitate high, long buildings of great strength.

2. To obtain the full benefit of large converters, a high pressure and great volume of blast are required, necessitating a very extensive and costly compressor plant.

3. In order to resist this high blast pressure, the lining material of the shells must contain a larger proportion of clay. This is apt to cause a leakage of blast.

Weighing the advantages of large capacity against the drawbacks just pointed out, it is probable that a shell with a diameter of 6½ feet, and a height of from 11 to 12 feet (outside measurements) combines the most advantages. Such a shell will weigh, when lined, about 34,000 pounds, and when running on 5 per cent. matte, will pour an initial charge of about 2,800 pounds, and a maximum charge of 10,000 pounds of blister copper.

A Chili mill, capable of grinding and mixing for two converter-sets, such as already described, will prepare the material for linings. In case it should be overcrowded, a pug-mill may be added for the mixing of the quartz and clay, while the Chili mill is retained solely for grinding the clay. The pan of this mill, with false or replaceable bottom, revolves, while the rotating rollers remain stationary. This mill is driven by a separate small engine which also operates the crane, and the hydraulic pump for tilting the converters.

The blast is furnished by a horizontal compressor. This consists of a compound, condensing Corliss engine, with steam-cylinders respectively 18 inches and 32 inches in diameter, and a 42-inch stroke. The air-cylinders are 42 by 42 inches. To supply a three-converter set (two converters constantly running), at an average pressure of 11 pounds per square inch, the compressor must furnish 3,000 cubic feet of free air per minute. To effect this, it must make from 40 to 45 revolutions and develop about 100 horse-power. This engine, if required, is capable of supplying four converters, running simultaneously, with sufficient blast at a pressure of 9 pounds; or five converters, at a pressure of about 7½ pounds.

The capacity of the converters depends largely upon this pressure. With the type of converter used at the Parrot smelter, a pressure below 9 pounds would cause the work to go too slowly, and also necessitate much tuyere punching, while anything exceeding 12 pounds pressure per square inch will hurl too much molten material out of the vessel. The deep, modified Parrot converters at Great Falls easily stand a 16-pound blast.

During the operation of turning a converter-body up or down, several of the rear tuyeres will evidently be left completely bare and free from the molten bath, while a correspondingly deeper layer of matte will be thrown above the front ones. This is the moment at which the "filling" of the wind-chest with matte is most likely to occur, for the greater proportion of the blast naturally streams out of the uncovered tuyeres where it meets with no resistance, while the deeply-buried openings, that need the greatest blast-supply, oppose the most resistance.

Hence, there must always be a large surplus of blast to guard against such mishaps. Furnishing the blast is the most expensive item in the ordinary bessemerizing of copper.

We now have the data for estimating the amount of power required for a three-converter plant of the type under consideration. With two converters constantly running at a blast pressure of 11 pounds per square inch, and turning out some 65,000 pounds of blister copper per 24 hours from 55 per cent. matte, there will be required:

For air-compressor	100	horse-power.
For Root blower (remelting cupola, etc)	15	"
For Chili mill, crane, and hydraulic	10	"
Total	125	"

COST OF A THREE-CONVERTER PLANT OF THE TYPE DESCRIBED.

It is impossible to give an accurate estimate of the cost of a completed plant of this description, as the total will depend much upon the solidity of the buildings, the extent and elaborateness of the dust-chambers and flues, and, above all, the general configuration of the ground. The following estimate can only pretend to give the cost of the main items (at Butte, Montana), leaving the details to be filled in according to local wages and conditions. The setting of the engines and blower, the dust-chambers, stack, and building are thus omitted.

Air-compressor as described, complete	$10,000
Root blower, No. 6	1,400
Small engine for Chili mill and cranes (20 horse-power)	600
Plunger pump and hydraulic accumulator	1,200
Sixteen-ton crane	2,000
Chili mill	800
Seven complete converter shells, and 2 extra tops	3,800
Standards, moving machinery, cylinders, trunnions, etc.	6,000
Two bowl buggies	950
Three copper-mold buggies	525
Cupola furnace, complete	1,200
Total	$28,475

Though the first cost of a good Bessemer plant is large, the actual cost of converting copper is quite moderate, being usually below three-fourths of a cent per pound of copper. The product is also obtained in excellent condition for shipment to some central refinery where cheaper fuel and wages are obtainable than in most of our mining districts. It is to this process that the development of the copper mines of the Northwest are largely due, for there, above most other localities, expensive labor, fuel, and materials make large capacity an absolute *sine qua non* of success.

CONVERTER PRACTICE.

In France, Italy, Norway, and some other European countries, mattes containing only from 15 per cent. to 35 per cent. of copper are profitably bessemerized, the operation being divided into two or more stages. The converter linings stand from seven to ten blows, and there is no doubt that, if conditions should demand it, similar low-grade mattes would be bessemerized in the United States.

In ordinary American practice, where the cost of wages, fuel, and supplies are very high, and where a very large and constant output is demanded by the owners of the enterprise, it is found most advantageous to so classify the ores and conduct the preliminary roasting process as to produce a matte of 50 per cent. to 55 per cent. copper at the first fusion. This may be accomplished either in blast-furnaces or reverberatories, depending upon the mechanical condition of the ore and the composition of its gangue. This matte is charged into the converters in a molten condition, and subjected to the action of a powerful blast of air. Sulphur and iron begin at once to oxidize, both producing heat. The sulphur escapes as (mostly) sulphurous acid gas, while the iron, being changed to ferrous oxide, combines with silica taken from the acid lining of the converter, also producing heat. When the iron is removed, the slag is poured off, leaving a nearly pure cuprous sulphide, or high white metal. After this momentary interruption, the blowing is resumed until the remaining sulphur is burned off, and only metallic copper (containing silver and gold in the majority of cases) remains behind. This blister copper is now poured off into iron molds, in the shape of pigs, or anode-plates for electrolysis, and the converter is ready for a fresh charge of matte.

The original converters introduced at the Parrot works in 1884 are shown in Plate X. They were very expensive, being constructed entirely of wrought iron, with the exception of the cast-iron blast ring, shown separately in the illustration. They were not removable from their stands, and, when the lining required patching, had to be cooled with water, so that only one of the set of three could ordinarily be kept in steady operation. At first, they were burned down and raised by a crank actuated by hand; later, steam-power was used for this purpose. They are now only of historical interest.

In 1889, cheaper cast-iron bowls (shown in Plate XVI.) were substituted. These were made replaceable by hanging them in stands, and were tilted by hydraulic machinery similar to that shown in Plate XIII. These bodies were very heavy and subject to much repair and leakage of blast from cracking. Their best point was, that the lining could be run until it was very thin without much danger of burning the shell.

The present still less expensive, and very effective wrought-iron shell with cast-iron trimmings, shown in Plates XIII. and XIV., was introduced by Keller in 1891, and gives excellent satisfaction. Modified shells of the same pattern, but of larger size, are now used at Great Falls and Anaconda. Accumulated experience points to the employment of these large converters wherever circumstances will permit. In proportion to their larger charges, they require less relining, and, since a higher blast pressure can be

used with them they run proportionately faster. The flame reactions, however, by which the progress of the operation is mainly judged, become less and less distinct with the increase in depth of the charge.

Without going into the history of the numerous experiments and discarded apparatus that have been tried, it is intended to describe the most approved copper bessemer-practice as now executed in the United States. Unless otherwise specified, all figures refer to the last Parrot converter plant, shown in Plates XI. and XII.

Perhaps no metallurgical operation requires more experienced workmen, or more careful supervision and watching all along the line, than does the converter process. The remelting cupola must run steadily; the converters must not be delayed, nor must they be burnt, yet the last fraction of duty that they are capable of must be got out of them before relining; great judgment must be used to prevent fatal breakages of the crane; the linings must be well put in and dried, and fresh converters must always be on hand to replace the corroded ones. To keep all these operations in accord, and to lose no time or charges, requires great ability, zeal, and experience on the part of the foreman.

The following tables (II. and III.) illustrate the results obtained by the operation which we are about to describe. The weights and assays given are accurate, but it is impossible to calculate the metallurgical losses from these results, there having been no complete clean-up and stock-taking at the moment corresponding to the termination of these periods.

TABLE II.—RESULTS OF BESSEMERIZING COPPER MATTE.
Running time, 333¼ days.

	Weight in Pounds.	Assays.		
		Cu, Per Cent.	Ag, Ounces Per Ton.	Ag, Per Cent.
Coke used for remelting matte	3,814,248	....	....	....
Matte charged	....	55.4	61.9	0.213
Blister copper produced	18,622,700	99.1	114.44	0.393
Chips for resmelting	1,138,480	7.5	9.00	0.031
Flue-dust from cupola	453,610	33.9	33.00	0.113
Flue-dust from converters	154,500	64.9	65.7	0.225
Converter slag	....	1.16	0.60	0.002
Cupola slag	....	0.56	0.78	0.0027

Table III. is the result of one month's (31 days) steady running. and is typical of the year's work. The ores smelted at the Parrot works contain considerably more silver than the normal ores of the Butte district.

TABLE III.—RESULTS OF BESSEMERIZING COPPER MATTE.

Running time, 31 days.

	Weight in Pounds.	Assays.		
		Cu, Per Cent.	Ag. Ounces Per Ton.	Ag. Per Cent.
Coke for remelting matte..............	305,234	....	....	....
Quartz for lining converters............	400,000	....	....	....
Clay for lining converters	120,000	....	....	....
Matte charged..........................	....	55.9	59.0	0.200
Blister copper produced................	2,075,609	99.2	103.84	0.356
Chips for resmelting (75.4 per cent. SiO_2)	53,980	10.0	9.0	0.031
Flue-dust from cupola..................	33,610	38.8	42.5	0.146
Flue-dust from converters..............	3,810	54.8	55.0	0.190
Converter slag*........................	....	1.1	0.4	0.0014
Cupola slag............................	....	0.5	0.6	0.002
*Reduced by retreatment to............	....	0.3	0.3	0.001

According to Dr. E. Keller, many copper mattes correspond to the formulæ $x\text{Cu}^2\text{S} + y\text{FeS} + z\text{Fe}^3\text{O}^4$, or $\text{Cu}^2\text{S} + y\text{FeS} + n\text{Fe}^2\text{S} + z\text{Fe}^3\text{O}^4$. His opinion is based on a large number of varied analyses, extending over a series of years. The following are some of the most typical:

PARTIAL ANALYSES OF COPPER MATTES.

	Cu.	Fe.	S.
1. Parrot reverberatory matte...	29.41	34.38	23.70
2. " " "	36.15	31.13	23.88
3. " " "	38.52	30.00	22.61
4. Baltimore cupola matte	46.85	... .	22.34
5. Boston & Montana reverberatory matte...	52.61	21.65	22.12
6. " " " "	56.92	17.90	23.80
7. Anaconda matte......................	60.22	12.12	22.58

The matte to be treated in the converters averages about 55 per cent. and comes from both reverberatory and blast-furnaces. After having been broken into lumps not exceeding 12 pounds in weight, it is delivered on to the charging-floor of the remelting cupola. The function of this furnace is simply to provide a sufficiency of molten matte for the converters. The proportions of matte and coke charged are determined by the cupola-man, the matte requiring to be heated considerably above its melting point in order that it may be sufficiently fluid to run through the long spout into the converter. The coke used for this remelting of the matte contains about 17 per cent. ash. The weight of the coke used is about 8.2 per cent. of the matte remelted, as seen from Table III. (The slightly larger proportion of coke shown in Table II. includes a certain quantity used in drying the converters, a plan that is no longer practised.)

It is a somewhat curious fact that, while in smelting the sulphide ores in blast-furnaces for the matte already described, the coke used varies from 7.5 per cent. to 10 per cent. of the charge, according to its quality and percentage of ash, in the remelting furnace the varying percentages of ash in the coke seem to have no influence upon the proportion of this material required, providing only that it is firm and massive. As a rule, nothing is added for fluxing purposes. A small addition of low-grade matte, or heavy sulphide ore, is sometimes added when the charge is too rich in copper; or, high-grade matte may be added to raise the tenor of the product for the converters. Metallic chips or scrap copper should never be added to the matte charge, as it is apt to sink to the bottom without changing its composition, and build up a chill on the sole of the crucible.

By using less coke for this operation, and, especially, in combination with a heated blast, more of the oxidizing (pyritic) effect would be obtained during this remelting. This would, naturally, cause a greater concentration in the grade of the matte during its passage through the cupola. At present, the enrichment seldom exceeds 3 per cent. in copper, and this is mainly due to the elimination of small quantities of slag that may have adhered to the matte, and to direct reduction by carbon. The heat generated by the oxidation of the matte, as just suggested, would be much lower than that derived from coke, since the oxidation of sulphur generates only about one-fourth as much heat as coke. This might still be ample to produce the required temperature for the liquid matte, and would also avoid reducing any copper to a metallic state.

As already intimated, this crucible, or well, of the remelting cupola is lined with quartz containing but little clay, as this lining suffers but slightly from the blast pressure. The bottom is put in only 2 inches deep; merely to protect the sole-plate, and to form a tight joint between plate and ring. The tap-hole is protected with a few fire-brick. The sides are made only 4 inches thick at the bottom, and increased but slightly until near the top of the crucible, when the lining is quite suddenly thickened to 14 inches, where the principal wear and corrosion occurs. In about five weeks the lining is worn so thin that the well has to be changed. It also has a tendency to chill on the bottom, thereby gradually raising the tap-hole and decreasing the storage capacity. The extent of these annoyances depends mainly upon the care and skill of the workmen. The most important precautions are to feed the cupola steadily, using the smallest possible proportion of coke consistent with the desired results, and to keep the tap-hole as close to the bottom as possible. This is effected by inserting a steel bar into the tap-hole as soon as it has been plugged from the preceding tap, and gently driving it forward, ready for the next tap. It is also necessary to let the well run out dry occasionally, in order to blow out any fine coke that may have settled on the bottom of it.

A small amount of basic slag is formed in this cupola. It is of a very variable composition, being partly due to the ash of the coke, partly to the slag that may stick to the lumps of matte charged, and partly to ferrous oxide produced by the blast. When the well is full, the thin slag is drawn off the surface of the matte, taking care to get it as clean as possible, as it is too poor for re-treating, its daily sample for many months having averaged only 0.5 per cent. copper and 0.75 ounces silver to the ton (0.0025 per cent.). The cold slag-cones are broken on the edge of the dump, and the tips sent to the ore-blast-furnaces.

Metallic bottoms of copper, in this cupola, are usually a sign of faulty work, but they cannot always be avoided. When formed, they are usually fed into the ore-blast-furnaces, where, with zinc-bearing ores, they sometimes form an alloy resembling brass, found, on cooling, as nodules enclosed in the matte. The following are average analyses of metallic bottoms, of the brass alloy just referred to, and of three monthly samples of matte from the ore-smelting furnaces:

	Cu.	Fe.	SiO_2.	S.	Zn.	Pb.	Mn.	Ag.	Sb.	Oz. per Ton	
										Ag.	Au.
Bottoms	79.88	13.95	......	0.50	2.70	1.09	0.36	0.18	0.69	163.5	7.94
Brass	62.03	0.11	......	......	36.99	......	......	......	......	6.6	......
Blast-furnace matte.	49.48	22.45	......	24.55	......	0.95	......	......	......	49.5	......
" " "	50.23	20.20	0.66	23.32	2.37	2.80	......	......	......	52.40	0.12
Reverberatory matte	59.11	12.62	0.81	22.94	2.18	1.73	......	......	......	72.90	0.59

It takes a crew of three men per shift (12 hours) to attend such a cupola on a regular duty of something over 60 tons of matte per 24 hours. These consist of a feeder, at $3.75; a tapper, at the same wages; and a helper, at $3.

The loose quartz from the old linings of abandoned wells is used for lining converter tops. The remainder of the lining, consisting of a mixture of silica and matte, is broken small and either re-fed to the same cupola, or used in the ore-furnaces.

The remelting cupola makes about one per cent. of flue-dust, which, being due entirely to mechanical causes, corresponds in grade with the matte charged.

The remelting cupola is an expensive and superfluous portion of a modern Bessemer plant, but has to be put up with in cases where the works are not laid out to admit of the direct tapping of reverberatory smelting furnaces into the converter, or the employment of reverberatory forehearths for the ore-blast-furnaces.

The converter department next engages our attention, and we will follow the new shell-sections from the boiler shop until they enter active service. The two lower sections are put together with the aid of the crane, some asbestos packing being placed between the flanges, and the bolts being tightened. Next, the bottom lining is tamped in, consisting of a mixture of 83 per cent. quartz and 17 per cent. plastic clay, by volume. This is put in to within 6 to 8 inches of the tuyeres (which are originally about 20 inches above the bottom-plate), according to the grade of the matte under treatment. Then a tapering iron tub is placed in the converter-bowl to form the interior cavity. It is 48 inches high, 36 inches in diameter at the top, and 28 to 32 inches at the bottom, according to grade of matte. Around this pattern, the lining mixture already described is firmly tamped by the aid of iron bars, kept hot by occasional immersion in molten slag.

When the lining has been tamped to the upper edge of the tub, the tuyere openings are punched through, and the tub hoisted out. The top section is now put on, and a workman enters the converter and completes the lining with balls composed of about 28 per cent. clay and 72 per cent. quartz, by volume. This mixture is usually made from old converter linings, with the addition of a little clay. It is necessary to use clay enough in this mixture to enable it to stick together into balls, and the adhesiveness of the lining is also much heightened by studding the inner surface of the tops with hooks. The tamped side-linings require less clay, yet enough to withstand the mechanical effect of the strong blast. In putting a perfectly new lining into a shell, still more clay is used in the mixture. This is done to make the lining adhere strongly to the iron wall, and thus prevent the blast penetrating between these two surfaces.

The lined converter is now placed to one side and dried. This is usually effected by throwing in a little wood and coke (nut coke will answer) and igniting it with hot slag, a very light blast being also admitted. It is of the utmost importance to have a sufficient number of converters so that the drying may be thorough and not too rapid. After this is accomplished, the converter-shell is ready to be placed in one of the stands for regular work.

It is then turned completely down, to dump out the wood and coke-ashes, this operation being assisted by the blast. Being turned up into an almost horizontal position, the cupola-launder is adjusted and the first charge of matte run in. The first charge will seldom exceed 2,500 to 3,000 pounds of matte, while, when the lining is burned thin, as much as 9,000 pounds is charged, the average not varying far from 4,000 pounds.

While charging, a light blast is turned on to prevent cooling. As soon as the proper quantity of matte has been received in the top of the converter, which is turned so low that the level of the liquid material does not reach any of the tuyere openings, the cupola well is plugged, and the spout removed from the mouth of the converter. Some cold chips (floor-sweepings, etc., containing much metallic copper and rich matte) may now be added, if desired. The converter is then turned up into its normal position, the full force of the blast being used to prevent the liquid charge from entering the tuyeres.

The charge now enters upon the first, or slag-forming stage of the process. With 11 pounds blast pressure, but little punching of the tuyeres will be found necessary during this period. The end of this stage is determined by the appearance of the flame issuing from the mouth of the converter. This part can only be learned by actual experience, and is difficult to describe. In a general way, it may be said that the greenish border of the flame gives way to a pale, permanent blue, when all the iron is oxidized.

The slag from this stage consists mainly of the ferrous oxide derived from the combustion of the sulphide of iron present, combined with the silica which it dissolves from the lining of the vessel.

These reactions may be imagined as follows:

$$FeS + 3\ O = FeO + SO_2 \text{ and } xFeO + SiO_2 = xFeO,\ SiO_2.$$

A matte low in copper quickly produces a thin slag; this is skimmed when the regulus is brought up to about 76 per cent. copper, and is thus nearly free from iron, else the slag soon becomes foamy and fills the whole interior of the converter. The great quantity of heat produced by the vigorous oxidation makes, curiously enough, the slag more siliceous when the matte is of low tenor in copper. The richer mattes make slag more slowly, and it is usually thicker and more ferruginous. This latter slag holds a much greater proportion of matte particles in mechanical suspension, which, however, is of slight importance if it has to be retreated anyway; while its large excess of iron makes it a welcome flux for the ore furnaces. In this latter case, as the slag does not foam and fill the vessel, the blowing is usually continued a little longer, and until the regulus contains some 80 per cent. copper, a small proportion of this metal being in a metallic condition.

A good, average charge makes 3 to 3½ pots of slag, at 450 pounds each, or some 1,500 pounds in all. Each of these slag cones consists of three different layers, viz.: a siliceous scum on top, then a deep layer of well-fused slag, and, at the lower extremity, an apex of matte, more or less mixed with slag. This slag crystallizes in very dark-green sub-translucent plates. A weekly sample of the slag resulting from the bessemerizing of a 55 per cent. matte, gave:

	Parrot Slag		Anaconda Slag. (Schnabel).
	With Acids.	Fused.	
SiO_2	41.90	36.80	35.70
FeO	48.06	50.40	55.83
MnO	....	....	0.22
Al_2O_3	....	6.80	1.76
CaO	....	trace.	....
ZnO	....	4.43	0.86
S	....	0.47	1.03
Cu	1.00	1.00	2.14
Ag	....	0.002 (0.7 oz.)	....
		99.80	97.54

The average assay of the clean converter slag for a year's run was 1.16 per cent. copper and 0.6 ounces silver to the ton (0.002 per cent.). This would hardly pay for resmelting were it not for occasional admixed shots of matte, and especially its coarse condition and its excess of iron, both of which circumstances render it invaluable as a flux in the ore-blast-furnaces. In Great Falls, this slag is poured, by the intervention of an electrically-moved ladle, directly upon a reverberatory charge just before the latter is ready for skimming. This settles all the matte grains, and makes the slag poor enough to be discarded at once.

Where this slag is used again as flux, it is, of course, advantageous to obtain it as free from silica as possible, both to augment its percentage of iron and to lessen the extraction of silica from the converter lining. This highly basic slag is apt to be very thick and "chunky," although a more siliceous slag may simulate this appearance if there is a lack of heat.

Bellinger gives the following determinations of such converter slags:

	Cu.	Fe.	SiO_2.
Liquid slag	0.7	43.0	37.8
Liquid slag	1.6	46.3	34.0
Thick slag (hot)	5.3	44.8	27.6
Thick slag (cold)	4.4	43.0	35.4

As in the remelting cupola, the addition of lime to the converter charge seems to have little or no effect in reducing its capacity for taking up the valuable metals. The following assays represent considerable quantities of slag:

	With Lime.	Without Lime.
Per cent. copper	1.4	1.6
Ounces silver, per ton	1.1 (0.0037%)	1.1

The flame from this first stage of the operation is dense, owing to the volatilization of zinc, lead, etc. It escapes in thick, white clouds, tinged slightly at the edges with green and red. In a very short time, however, these volatile metals have disappeared, and the white clouds become less dense, the red changes to rose, and finally fades out entirely, and the flame for a short period is pure green. As the iron in the charge becomes mostly removed by oxidation and combination with silica, a bluish border forms on this green flame, and this gradually spreads until the entire flame becomes of a uniform, pale bluish hue.

The slag is commonly liquid enough to be poured off from the turned-down converter; otherwise, it is drawn off with a rabble. This operation is called skimming, and is continued until copper sulphide is seen to lodge, as dark spots, on an iron bar or plate placed upon the slag-pot. A thin sheet of slag always remains upon the matte in the converter, and, indeed, is essential to keeping up its temperature by acting as a blanket during the remainder of the operation.

The greatest care must be exercised in tipping the loaded converter; otherwise, metal will enter the tuyeres, causing great delay and expense. As the matte flows gradually into the concave top of the converter as it is turned down, one tuyere-hole after another becomes uncovered, thus allowing an abnormal amount of blast to escape without penetrating any of the metal, while a corresponding decrease of wind-pressure occurs at the tuyeres that are still covered by metal. Ample volume and pressure of blast, and experienced men are needed to prevent mishaps. When all the tuyeres are uncovered, and the matte has collected completely in the now inverted top, the blast is nearly shut off.

Cold matte is now again thrown into the converter charge. This time it is done to furnish a little sulphide of iron to act as fuel and give a good start to the second stage of the operation. For a long time, coke or wood was added at this stage of the process, but this is now replaced by the ferrous sulphide of the cold matte, and also works up a large amount of chips, sweepings, etc., that would otherwise have to pass through the remelting cupola again. This low-grade matte also furnishes the iron for the formation of a thin slag-coating upon the rich matte in the converter, as already referred to.

The blast is again turned on, the converter raised to its normal working position, and the tuyeres quickly punched to break off the noses which have formed over them during the skimming operation.

The color of the escaping flame is now very variable, being greatly modified by the size and temperature of the charge. With a heavy or hot charge, the flame will be a mixture of blue and white, while if the same material be present as a light, or a cold charge, the flame will have a reddish cast. Under ordinary conditions, the flame is a bluish white, changing gradually into a rose red, and finally turning to a brownish red. It lessens in length and volume toward the close of the operation, until, at the last, there is only a brick-red flicker.

These end changes are too faint and uncertain to rely upon, and an inexperienced man can very easily go on blowing after all the sulphur is removed, and, indeed, until a notable proportion of the copper is oxidized, and the charge chills. This is a most unfortunate and expensive accident, and involves very serious labor in cutting out the malleable contents of the converter.

Fortunately, we have more certain, though still very slight, warning of the completion of the process. This consists in the projection of minute spark-like metallic globules against the rear wall of the hood, or dust-chamber, into which the fumes of the converter discharge. At first these sparks stick mostly to the wall, glowing brightly for an instant. But they soon come in greater quantities, and, losing their quality of glowing, rebound sharply from the wall. When the great proportion of them act in this latter manner, it is a sign that the charge is finished.

If a charge be decidedly overblown and contain too much oxide of copper, it can be rapidly brought back to the metallic condition by the addition of raw matte. But it is unsafe to run molten matte into an overblown charge, for the reaction between the sulphides and the oxide of copper is so violent that the whole charge may be blown bodily out of the converter. The proper way is to add raw matte in small lumps with a little slag, or sweepings from the floor, and a very few moments' blowing will put the charge into a proper condition for pouring.

Where it is desirable to make blister copper running over 99 per cent., it is necessary to always continue the blowing until a certain amount of oxide of copper is produced.

As the second stage of the operation proceeds, the molten mass gradually loses some of its heat; that is to say, the evolution of heat from the oxidation of the small percentage of sulphur still remaining in the charge is not sufficient to maintain the desired temperature. While this is going on, the blast rolls the thin sheet of overlying slag into numerous globules, their size depending largely upon whether the finish is hot or cold. In the former case they are small, seldom being larger than a bean, and comparatively uniform in size. With a lower temperature, they become larger, at times reaching the size of the fist. Each globule has a grain of quartz for its center, and next to it usually occurs some metallic copper. If the charge, during this second stage, should chill enough to endanger the operation, recourse may be had to wood, or, what is much better, the temperature may be restored by an additional tap of matte from the cupola.

The converter is now turned down ready for pouring. According to ordinary practice, a carriage containing a number of molds, and running on a track beneath the converter, is pushed backward and forward by man power. The resulting pigs, or bars, usually weigh from 215 to 275 pounds. The molds are made of soft cast-iron, or are cast out of copper from the converter itself, a wrought-iron plate being used for a common bottom, while only the sides are formed of copper. If the bottom of the mold were also made of blister copper, the bars would weld to it at the point where the stream of metal strikes it in pouring.

The converter is now ready for the next charge, and contains nothing but the granulated slag already referred to, which will serve as a siliceous flux for the next operation.

The first charge in a newly-lined converter is, necessarily, always small. It is, therefore, well to use a somewhat low-grade matte for the first charge or two when this is possible, and thus eat out the lining so as to make a larger cavity. This is decidedly preferable to leaving a larger cavity in the first place by the use of thinner linings, since it makes the corrosion at just the spot desired, and at the same time glazes and encrusts the whole interior of the cavity, thus effectually heightening its resistance to mechanical corrosion. Theoretically, of course, a converter-shell, widening toward the bottom and lined over a straight tub, would seem most desirable. But this is accompanied with such evident disadvantages of construction both in stand and shell, together with awkward lining, that they more than counterbalance any good that might accrue from it.

Large charges are, naturally, much the most profitable, and it is frequently tempting to run just one more charge on an already dangerously thin lining. It takes a very experienced and judicious foreman to get everything practicable out of a lining, losing no possible large charges, nor yet taking any grave risks of burning through the shell. When a lining has become dangerously thin, and an increasing redness of the converter shell threatens a catastrophe, the "pouring out" of an unfinished charge may frequently be avoided by cooling the overheated portion of the shell by a stream of water from the hose. Cast-iron converters cannot be treated in this manner, but these stand closer running, as regards thickness of lining, than do the wrought-iron shells. The pouring out of an unfinished charge causes considerable delay, and is usually a sign of poor judgment on the part of the foreman. Exceptionally, a charge is thus poured in an unfinished condition, when it has been introduced into a converter merely to "rinse out" the metal that may be clinging to the interior, and that would render the approaching relining more difficult. But, where possible, this should be poured into a ladle, and introduced at once into one of the other converters.

A lining is usually abandoned after the ninth charge. The granulated slag is dumped by turning the converter completely over, and the shell is then loaded upon the replacing buggy, which conveys it within reach of the crane. After the top is removed, the vessel is set one side to be air-cooled. Before the converters were made replaceable in the stands, it was attempted to save time by cooling them with water. This practice was not only injurious to the shell, but also cracked the lining so as to cause serious leakage of wind.

When the lower sections are sufficiently cool to work at comfortably, they are held in an almost horizontal position by the crane, and, with the aid of bars, are cleaned out sufficiently so that the new lining can make a tolerably firm and clean junction with the old remnant. The body is then placed upright and the relining continued in the manner already described, using, however, a somewhat smaller tub than is used for the original lining, as the material now being put in is not quite so firm, and must be made somewhat heavier. The size of the interior cavity is also influenced by the grade of matte to be run, it being made larger for rich mattes, where there is less corrosion. The proportion of clay in this *relining* material is also somewhat less, as the bond with the iron shell has already been made.

The built-up crust of slag and metal is now cut away from the cooled top, removing the snout during this operation, if more convenient. Before bolting it to its place on the converter, it must be perfectly cool, as a workman has to enter the vessel to complete the lining. For this reason, two extra tops are provided to each converter set.

With a low-grade matte, the lining of the tops is quickly eaten out. This is due to the washing-out action of the large volume of liquid, foaming slag. With a high-grade matte, on the contrary, the top is soon coated with heavy accretions. These have a tendency to close up the mouth of the vessel, and, in spite of repeated and prolonged chiseling, may cause the abandonment of a converter before its lining is used up. The object, therefore, is to steer a middle course between these two dangers, producing, at the same time, a large output of blister copper, and the least possible amount of slag, which shall also be sufficiently basic to make a good flux for the ore furnaces.

Linings, if properly put in, corrode with such uniformity that patching is rarely resorted to. When required, it is done with the converter in place, tilting it to the most convenient position and using the balls already described as employed for lining the tops.

Although a converter is relined, on an average, after every nine blows, thus lasting little more than a single 12-hour shift, it requires a completely new lining only about once in seven weeks. For the seven bodies, this would make one new lining to put in every week.

The following table gives the duration of the respective periods in converting a 55 per cent. matte.

	Slagging.	Making Copper.	
1	30 minutes.	30 minutes;	new charge.
2	45 "	40 "	
3	30 "	30 "	second charge.
4	35 "	45 "	second charge.
5	35 "	55 "	
6	30 "	35 "	second charge.
7	50 "	55 "	
8	45 "	55 "	
9	40 "	55 "	
10	50 "	50 "	3,250 pounds copper.
Average	39 "	44 "	

A charge that is at all cold will require an unreasonably long time for the second stage. For instance, a charge completed the slagging period in 35 minutes, but, running a little cold, did not come to blister until 90 minutes more had expired.

It is seldom necessary to tap more than one charge of matte from the cupola into the converter, for a single blow, except for the following reasons.

1. Where the converter lining is so corroded that the vessel will contain an exceptionally large charge.

2. When the finishing period becomes too long, and it is necessary to warm up the contents of the converter with fresh matte.

3. When the bottom of the cupola-well has become so raised by chills that it will not contain a proper charge of matte.

A three-converter set, under the conditions assumed in the preceding pages, will make 32 blows per 24 hours, averaging 32 to 33 tons (64,000 to 66,000 pounds) of high blister daily. At the Parrot smelter, the greatest output for any single 24 hours has slightly exceeded 38 tons (76,000 pounds).

The converter-gang required to accomplish this consists of the following men, most of whom work 12-hour shifts:

2 foremen at $5.00 (could superintend two or three such plants)	$10.00
2 skimmers at $3.75	7.50
2 liners at $3.50	7.00
2 cranemen at $3.50	7.00
10 laborers at $3.00	30.00
2 cupola feeders at $3.75	7.50
2 cupola tappers at $3.75	7.50
2 cupola helpers at $3.50	7.00
Total, per 24 hours	$83.50

The prevailing method of casting bars is not satisfactory or neat, though the spattering of the metal is not as shiftless as it appears, the cast-iron floor plates being easily swept up, and the metallic chips added to the next charge with little loss of time or money. During the operation of pouring, much ebullition of the metal in the molds occurs from the escape of sulphurous acid gas. This may prevail to such an extent with coarse copper (96 or 97 per cent.), that only a shell remains of an apparently filled-up mold, the bulk of the metal having escaped in the shape of a fiery rain, leaving the residual copper of somewhat higher grade than the original metal. Below this grade, bars are more or less spoiled by the presence of matte. A good bar is generally made up of several successive pourings. At 99 per cent., the surface of the bar becomes covered with excrescences produced during the process of solidifying. After cooling, the surface of the bar is covered with a thin film of black oxide of copper, unless it has been dumped into water while still quite hot. It then exhibits remarkably pure and uniform surfaces. Before shipment, each bar is trimmed with a hammer to prevent loss, and the metallic chips are placed in the molds to be embodied in the next cast.

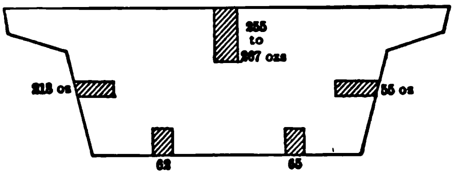
Though all three metals—copper, silver, and gold—are now in the best possible condition for shipment, the sampling of the bars presents a most vexatious question. This point becomes peculiarly important if they are to be sold in open market instead of going to a refinery controlled by the same smelting company. There are three methods of sampling now in vogue:

1. Securing chips from each bar by drilling or gouging them out. This sample, though cheaply obtained, is neither accurate nor uniform, and is usually practised only as a general guide in shipping to a branch refinery. The accompanying sketch, after Ray, of the Anaconda, illustrates the extreme irregularity of the distribution of silver in a converter bar. The figures indicate ounces of silver per ton of 2,000 pounds.

The resulting chips are melted and finely granulated by pouring into a long column of cold water, which makes a very uniform sample. An average assay of such chips yielded:

	Copper.	Silver. Ounces per ton.	Gold. Ounces per ton.
Before melting ...	95.60	84.56 (0.29 per cent.)	0.38 (0.0013 per cent).
After melting.....	98.97	87.58 (0.30 per cent.)	0.39 (0.0013 per cent.)

2. Taking a ladle sample while the converter is pouring serves also only as a rough check.



DISTRIBUTION OF SILVER IN A BAR OF COPPER.

3. The most correct way to obtain a sample is to take it from the reverberatory furnace when the converter bars are being melted for casting into anodes at the electrolytic refinery.

The following assays of large lots of Anaconda matte and blister copper from the converters may be of interest:

ANACONDA MATTE.

Copper. Per Cent.	Silver. Ounces per Ton.	Gold. Ounces per Ton.
57.17	58.80	0.32
55.93	57.34	0.34
58.39	56.00	0.40
56.13	52.70	0.50
57.12	55.14	0.46
58.55	57.18	0.54
56.16	51.60	0.46
56.26	51.86	0.42
55.90	52.50	0.30

ANACONDA BLISTER COPPER.

Copper. Per Cent.	Silver. Ounces per Ton.	Gold. Ounces per Ton.
99.17	87.50	0.42
99.18	90.30	0.52
99.05	84.10	0.60
99.10	82.60	0.63
99.15	80 40	0.68
99.17	79.80	0.77
99.18	81.10	0.83
99.15	79.40	0.62
99.15	80.00	0.66

The commercial assay for copper, silver, and gold is usually made by dissolving one gram of the sample in 5 to 10 c.c. of nitric acid of specific gravity 1.42. The solution is evaporated nearly to dryness, cooled, and 1 to 2 c.c. of concentrated sulphuric acid, together with sufficient water, are added. This solution is electrolyzed in the ordinary manner, the copper being precipitated on a platinum cone. From this weight, the silver must be deducted, and, for still greater accuracy in handling large quantities of material, allowance is made for arsenic, antimony, lead, bismuth, and oxygen, which substances are determined from time to time by exhaustive analyses.

For the determination of the silver and gold, a combined wet, and fire assay is commonly used. One assay ton (29.166 grams) of the sample is dissolved in 140 c.c. of nitric acid of specific gravity 1.42. After complete solution, the liquid is diluted to 500 c.c. and a slight excess of sodium chloride is added. After thorough stirring, it is allowed to stand over night; then filtered, and washed with hot water. The filter is dried and ignited with argols, the residue covered with a little litharge and one assay-ton of test-lead, and the ordinary course of the fire assay pursued.

If there is as much, or more, than one ounce of gold to the ton (0.00343 per cent.) in the copper, it is better to make a separate gold determination, in which the silver is only partially precipitated so as to avoid any excess of NaCl that might affect the gold.*

METALLURGICAL LOSSES IN BESSEMERIZING.

The fumes from the converters consist mainly of sulphurous acid and nitrogenous gases, carrying, besides the less valuable metals, a certain amount of volatilized copper and silver.

* For fuller details of these assays, we refer to Dr. H. F. Keller's paper on "The Analysis of Refined Copper," in the *Franklin Institute Journal* for 1894.

As in pyritic smelting, where most of the heat is produced by the oxidation of the sulphur and iron, it is found that the loss of gold by volatilization is exceedingly small under all conditions, while the loss of the other two metals is largely influenced by the quantities of volatile elements present, such as arsenic, antimony, lead, and zinc. To recapture these valuable vapors depends largely upon the provision made for catching the flue-dust. This at once brings us to the question of metallurgical losses, since we have already seen what becomes of the copper and slag that constitute the main products of this process.

The following table gives assays of two separate series of flue-dust samples, taken at two different works from the dust-chamber systems, starting at the converter openings, and proceeding toward the stack.

	Canal.		Dust-chambers.				Near Stack.
			Near Beginning.		Farther Along.		
Silver (ounces per ton)	28.0	64.0	21.8	65.0	19.0	46.0	17.5
Copper (per cent.)......	42.8	65.8	34.1	65.0	29.0	43.0	26.4
Lead " "	11.9	....	12.6	....	9.2	....	9.8
Zinc " "	6.7	....	5.2	....	8.7	....	10.4

These results seem to indicate that the longer the dust-chambers, the less will be the losses by volatilization. Experience has also shown plainly that with higher blast pressure, longer chambers are necessary.

It is probable that the most perfect and successful way to reduce the losses due to volatilization would be to filter the gases through canvas bags by means of exhaust fans, as is done in Lewis and Bartlett's process of smelting zinc and lead ores. Washing the gases scarcely seems feasible on account of the large quantity of free acid that they contain, and also, because this profound cooling would still further impair the chimney draught, already sorely affected by the cold air that enters the canals at the converter discharge.

From Table II., it will be noticed that a shipment copper that contains about one ounce of silver to the per cent. of copper, produces a converter-dust that is very similarly constituted. Eighty-one thousand pounds of blister, averaging 200 ounces silver to the ton (0.687 per cent.). produced dust assaying 62 per cent. copper and 99 ounces silver (0.34 per cent.). The converter dust, being all derived from actual condensation of volatile products, is very small in amount.

From the nature of the process, with its metallic by-products, it is evident that accurate sampling is practically impossible as a concomitant of ordinary running. The best criterion as to metallurgical losses is a series of long, continuous campaigns. Although such extended runs do not admit of sharp cut-offs and thorough clean-ups, their errors per ton are reduced by large production. Counting that the discarded slag will contain 0.3 per cent. copper and 0.3 ounces silver per ton (0.003 per cent.), and that the accumulated chips and flue-dust will, when discarded, assay the same (for the slag produced from their retreatment carries the same as the discarded slag just referred to, being indeed, part of it), the loss in converting, with good dust-chamber capacity and not too many volatile metals in the matte, amounts, under the conditions we have been describing, to from 1 per cent. to 1.5 per cent. of the copper treated.

The loss on the silver contents is somewhat higher, falling somewhere between 2 per cent. and 2.5 per cent.

Exceptionally, tin, cobalt, nickel, and bismuth are contained in the mattes. Of these metals, tin and cobalt are pretty thoroughly slagged in the converter. As would be expected from its behavior in the refining furnace, nickel remains longer with the copper; but, if the charge be slightly overblown, the last fall of copper-oxide slags shows a percentage of nickel entirely out of proportion to the amount present.

Bismuth is the most harmful of all the metals enumerated, owing to the tenacity with which it clings to the copper and the extraordinarily injurious effects that it produces, even when present in the most minute quantities. Hampe found that even 0.02 per cent. of this metal causes copper to break distinctly red-short.

There are two points in relation to converter losses that seem pretty firmly established: These are:

1. The loss by volatilization increases with the blast pressure (though, no doubt, this loss, whether great or small, can be recovered by proper and sufficient condensing apparatus), and is insignificant until the sulphur of the matte is so far removed that metallic copper begins to form.

2. This loss is greatly augmented by the presence of the volatile metals already referred to. In treating "concentration-matte" from lead-silver blast-furnaces with some 40 per cent. copper and

15 per cent. lead, the loss in silver was something incredible. During experiments of this nature on a large scale at certain works, I am informed, on undoubted authority, that the loss in silver by volatilization reached the preposterous figure of 50 per cent. of the total silver in the matte. The tools and neighboring iron work were covered with a thick rose-colored coating, carrying a high percentage of oxide of silver.

COST OF CONVERTING COPPER MATTE.

Under the conditions that prevail at, or near, Butte, Montana, which is the great center of this branch of American industry, the cost of converting copper matte during the year's run, 31 days' results of which are given in Table III., page 552, was, per ton (2,000 pounds) of matte, about as follows:

(The expenses of administration, and such general items, are omitted, as they vary according to the policy of the individual works.)

COST OF CONVERTING COPPER MATTE AT BUTTE PER TON OF 2,000 POUNDS.

Labor	$2.75
Power	1.50
Fuel (remelting cupola)	1.40
Supplies	1.10
Repairs, etc	1.25
Total	$8.00

The details of cost will be found distributed throughout their respective departments. This total is considerably reduced by the abolition of the remelting cupola, the employment of larger converters, and the substitution of water-power for coal. Six dollars per ton of matte will fully cover the cost of bessemerizing under the improved conditions just referred to.

The treatment of converter slag, under ordinary circumstances, costs practically nothing. Its value as a basic flux, and as a mechanical agent in loosening up the ordinary fine and compact charges of the ore-blast-furnaces will fully compensate for the slight amount of extra coke (usually, none at all) required to smelt it. But in cases where the cost of its rehandling exceeds these advantages, it will usually be more profitable to make the converter slag as clean as possible, and, after careful sorting, to put it over the dump. The cost of smelting will thus be reduced at the expense of a slight metallurgical loss.

MISCELLANEOUS.

Converter Linings.—This is one of the most important and unsatisfactory items of converter practice, and warrants a more detailed consideration than would be suitable in a practical description of ordinary running.

Since the ferrous oxide, formed by the combustion of the matte, is forced to steal its silica from the converter lining, it would seem more sensible to supply the same in the shape of quartzose ores, or even barren quartz sand, rather than to suffer the rapid destruction of the lining. But this apparently simple matter has hitherto proved impossible in regular work.

One of the earliest trials after the introduction of the process was to feed fine sand, or crushed quartzose ores into the blast pipe in such a manner that it was carried through the tuyeres into the molten matte charge. Contrary to expectation, this plan did not answer at all. Unless the sand-stream was cut off a long time in advance, the workmen punching the tuyeres had their eyes filled with it whenever the plugs were removed from the holes in the wind-chest; and, even if this serious annoyance could have been withstood or surmounted, it would have been of no avail, as the silica rose at once to the surface of the bath, where it aggregated into infusible lumps that floated about without dissolving in the slag to any appreciable extent.

Many other methods of supplying the necessary silica to the charge have been tried, such as throwing sand or siliceous slag into the converter, and even running the latter into the vessel, in a molten condition. But hitherto any slight saving of the lining has been far more than outweighed by the extra expense, delay, and inconvenience attendant upon these attempts to spare it.

Lining, therefore, is an arduous, expensive, and constantly recurring task, and metallurgists are constantly trying to hit upon some expedient to save time and costs.

A basic lining would at once suggest itself, either ground, and mixed with tar, or put in with magnesite bricks. A lining of Grecian magnesite brick was given a thorough trial. The bricks had the following composition:

SiO_2	1.60
CaO	2.05
$Fe_2O_3 + Al_2O_3$	1.61
MgO	94.96
	100.22

It was laid in mortar of similar composition. The two lower sections of the converters were lined with these brick, various plans of construction being adopted. First, the lining of these sections was made entirely of the brick. Then, to partially prevent radiation of heat, a thin layer of the ordinary acid lining was placed next to the iron shell, the brick forming the interior of the cavity. This order was then reversed, and the thin acid layer was placed next the matte in order to supply a certain amount of silica for the slag. Lastly, in order to strengthen the bottom, that part alone was formed of magnesite brick. None of these trials resulted successfully for several reasons, among which the three following were the most obvious.

1. The conductivity of the magnesite was so great that the exterior of the converter became dangerously hot, while the interior was correspondingly cooled, the result being that it took four hours to convert a charge that would normally be completed in 80 minutes.

2. As already explained, serious difficulty was experienced in adding silica in such a manner that it would combine with the ferrous oxide of the matte.

3. The exposure of these brick to great variations of temperature soon caused them to shell, and thus induced leakage of blast.

Water-jacketing the converter is also a favorite idea. One might fear that the heat would be transferred to the water with such rapidity that it would be impossible to keep up the required temperature in the vessel. But it is possible that a crust of matte would form upon the cooled, iron walls of the converter until it attained such a thickness as to prevent any undue loss of heat. When the crust became too thick, the charge in the converter would experience a rise in temperature, and a portion of the crust would be dissolved away by the molten matte, the thickness of the crust thus regulating itself automatically.

I cannot, however, believe in any such comfortable condition of affairs. Assuming even that the crust of chilled matte became sufficiently thick to prevent much transmission of heat to the cooling water outside, there would then be in the converter, and constituting its lining, a thick mass of high-grade matte (for it would be the rich matte that would form the chill), of such magnitude and such extraordinary conductive qualities that it would absorb heat from the molten metal until it (the lining) either was fused itself, or, as would probably occur, the entire contents of the converter became solid, not being able to generate heat as rapidly as it would be absorbed by this metallic lining. In other words, owing to the high conductive power of the metallic lining, there will always be a strong tendency toward the establishment of an equilibrium in temperature between it and the molten matte in the center. And, as the weight of this self-formed metallic lining will very many times exceed that of the liquid contents of the converter, while the amount of heat at our disposal, even with a non-conducting lining, is seldom much in excess of our requirements, it would appear to be impossible to keep a charge in a molten state under the conditions just described.

Those who doubt the correctness of this reasoning have only to recall the behavior of white metal, in a reverberatory furnace, when "roasting" for blister, or pimple metal.

The heavy pigs of white metal that are piled on the hearth of the furnace remain unaltered until their melting point is reached, when they begin to drip from every edge and angle. If the greatest care is not now taken in firing, and if the temperature is allowed to rise even a few degrees, the pigs flatten out all at once; not simply melting rapidly from the outside, as would a lump of ice under similar conditions, but collapsing *en masse*, as though they were mush.

The material that is universally employed for converter linings in the United States is a mixture of quartz and clay. The quartz is the desired substance, the clay simply being a necessary evil to hold the quartz grains together, that they may have sufficient cohesion to withstand the blast, as well as the surging of the melted metal.

A complete lining at the works of the Parrot, or the Montana Ore Purchasing Companies, at Butte, weighs, according to Bellinger, 2,600 pounds for the top, and 7,000 pounds for the two lower sections, making a total for each converter of 9,600 pounds. Experience in America has taught that the quartz used should be very pure, nothing below 98 per cent. silica being at all satisfactory or durable. For this reason, it is the universal practice to purchase pure, barren quartz rather than to make use of highly siliceous ores, even though the latter can be obtained at a very favorable margin. After experimenting with quartz crushed to all sizes, from that of an egg down to almost flour, it has been found that the quartz rock should come in lumps, which should be crushed to just a five-eighths inch punched trommel, and then be used as it comes, coarse and fine all together. Experiments conducted with quartz that had been classified into different sizes by means of graduated screens, gave very poor results as regards duration of the linings. The fine particles of quartz seem to be needed to fill the interstices between the coarser pieces, and when this filling is missing, the lining lacks cohesion and is easily washed away by the metal.

This occurs especially around the tuyere orifices, or at any point where unprotected edges exist.

The clay used for binding the particles of quartz together differs at the various works as much as do the opinions of the superintendents regarding it. While some men demand a good quality of fire-clay and claim that it is essential to the durability of the linings, others are willing to use any fair, plastic clay, holding that the manner in which the lining is rammed in is of much greater importance than the quality of the clay used. Judging from actual practice with a great variety of clays, it would seem that the less clay we can get along with the better the results. Above all things, the clay must be plastic, while infusibility, though desirable, is of secondary importance. At Butte, the best results are being obtained by the use of an extremely sticky clay of inferior quality, and such as may be found in almost any district. It has the following composition:

	No. 1.	No. 2.
SiO_2	69.70	64.02
Al_2O_3	15.20	16.02
Fe_2O_3	7.36	4.90
MgO	0.38	trace.
CaO	trace.	2.39
NaO	4.11	
KO	2.63	
H_2O	0.66	
	100.04	

Both quartz and clay are dumped into large bins direct from the railway cars. The clay must be carefully preserved from freezing, else it will be rendered practically useless for the intended purpose. In cold climates, this is usually attained by steam radiators, or by locating the clay bins near some flue that is always in use.

The quartz, after having been crushed to the proper size by means of a jaw crusher and Cornish rolls, is mixed in the desired proportion with the clay, which has already been crushed in the Chili mill.

As already intimated, two different mixtures are used in lining the converter. The lower sections are tamped with a rather dry mixture containing 10 or 12 shovels of quartz to 2 shovels of clay, while the top has to be lined with a much damper mixture, made up into balls, and usually consisting of old, crushed lining, to each 10 or 12 shovels of which are added 2 more shovels of clay.

To prepare and put in the linings, for a run similar to that exhibited in Table III., will require three men (one liner with two helpers) per shift of 12 hours. The ramming of the lining with heated bars demands a peculiar knack that is easier to acquire than to explain. If the quartz is too fine, or the men use their bars with a jumping, "springy" blow, the lining also becomes elastic and unsound instead of settling into the firm, solid mass desired. Such a lining will be quickly washed out by the metal and blast, and destroyed. To obtain a perfect lining requires a solid, dead stroke, the rammer pressing down, as it were, for an instant after the blow is delivered, and never rebounding.

The more slowly and thoroughly a lining is dried, the longer will be its duration, and it is for this reason that a set of 3 converter stands requires 7 converter-bowls with 2 additional tops, provided the plant is to be run to its full capacity. This drying is done with a light blast in connection with wood or coke.*

Of the seven bowls required for a three-stand plant, three are in use; two are usually drying; one is being relined, and the seventh is cooling preparatory to relining.

The trimming and cutting away of the old lining and adherent metal, in order to expose a fresh surface that the new lining may adhere to, is a tedious and difficult operation. Much of it is baked to a granite-like hardness, while it is saturated with metallic copper and matte.

* There is little doubt that the work can be done more neatly and expeditiously, and the fuel applied to very much better advantage by using the latter to heat the blast, and then drying the linings with a gentle current of hot wind. Desiccation is accomplished not so much by stagnant heat as by a current of dry air which removes the moisture as fast as it is evaporated. A box containing a suitably proportioned steam-coil would heat the air required for drying the lining, with the expenditure of something less than 25 per cent. of the fuel now used in direct drying.—E. D. P., Jr.

When the old shell of lining is completely removed, the débris is divided into three portions; the first, consisting of fused lumps, is sent to the remelting cupola or the converters; the second, consisting mainly of small, caked stuff, called "chips," generally goes to the reverberatory smelting furnaces; the remnant of dry linings is mixed with a little more clay and used for relining the tops.

CHAPTER XVIII.

THE ELECTROLYTIC REFINING OF COPPER.*

IF ONE circumstance more than any other were proof of the commercial success that has attended, in this country, the electrolytic treatment of argentiferous blister copper, that circumstance would be found in the enlarging of present electrolytic establishments and the construction of new works. At the present writing, plans are being prepared for two large refineries to be located in the East, while another, to be located west of the Missouri River, is under contemplation. This construction of new works is especially noteworthy now that competition has reduced the cost of treatment to a point where producers would rather pay for outside treatment than incur the expense of building their own refineries. But the very fact that electrolytic establishments are earning dividends, even with present charges for treatment, is the real incentive for additional works. And it can be said that when the price of silver shall have risen so as to stimulate copper mining and make the treatment of lower grades of pig copper more profitable to the producers, we shall see a still more pronounced activity in this direction, and consequent lower prices.

This success has been achieved not by any brilliant discovery that has revolutionized electrolytic methods—as the converter process has done for a different branch of the metallurgy of copper—but simply by the extension and amplification of known and tried formulæ. There have been, it is true, many departures from the old multiple system: as in the various "series" processes that have been established in this country by Hayden, Snow, Smith, and Stalman. Mention might be made also of modifications introduced by Mr. Thofehrn in making cathodes by a rotary surface in connection with a reciprocating spraying device, of a new method of preparing copper oxide for the maintenance of the

* Kindly prepared for this work by Maurice Barnett, S.B., Met. E., of Philadelphia.

electrolyte, of oxidized anodes, of solid frames for supporting simultaneously all the electrodes in a tank, and an electric crane for raising and removing them for re-preparation. It is somewhat doubtful if all these innovations have marked advances in metallurgical practice. The general consensus of opinion would seem to indicate that they had not. The series system seems to be losing ground. As to the use of a complex arrangement for handling copper in and out of tanks, that seems to have been abandoned and a return made to the old method of direct handling. Indeed,it is to the simplification of the electrolytic idea, combined with the most thorough adaptation of plant to its location and accompanied by the closest scrutinizing of the refining operation, that we look for the real means of cheap production of high-grade electrolytic copper.

While there are many so-called processes of electrolytic refining, all are ultimately reducible to two systems; and according as the electrodes in each are in "multiple arc" or in "series," they have been known as the "multiple" and "series" system respectively. As these various processes have been described in the technical journals, no specific mention need here be given. Of greater moment at this time is a consideration of the principle of relative value of the two systems in which all processes are included. In speaking of the "multiple" system, an arrangement is contemplated where the electrodes are in parallel arc, the tanks being in series; while in speaking of the "series" system, there is intended an arrangement, where the electrodes are in series, the tanks being either in series or multiple-series as the manager elects to have them.

In considering this question of the relative value of the two systems, the features that are of special moment to those interested are:

1. The efficiency of each.
2. The amount of copper held back in each.
3. Relative cost of operating each.
4. Relative cost of construction involved in each.

From a consideration of electrical principles it is obvious that the voltage required per tank will depend, neglecting the constant of solution, upon the area of the plates, the distance of the plates apart, and the number of plates in series. In a "series" tank therefore, the electromotive force must be many times greater than the electromotive force required in the "multiple" tank; and where many plates are arranged in series, the electromotive force required to maintain a given density of current is very great and leads to short circuiting, with consequences that may be serious. In an electrolytic tank, the slimes, as they collect on the bottom, form a large conducting plate. If the electromotive force is sufficient to overcome the resistance between the bottom of the electrodes and the layer of slimes, the current will flow partly through the electrodes and partly through the slimes—in amount inversely as the resistance of the two routes. The consequences, obviously, owing to the lessened density of current at electrodes, will be manifest in a diminished output per tank. That this short-circuiting occurs, is proved by the heavy deposit of copper noticeable at the end electrode in the series; for here the currents will recombine, producing a greater density per square foot of depositing surface than elsewhere in the tank. Another proof is the tendency to concentration of copper in the electrolyte of series processes, caused, most likely, by the fact that the copper falling to the bottom of the tank in granules, or sections of the anodes, forms, with the slimes, a large anode-surface not contemplated in the arrangement; and besides the ordinary corrosion of this copper, there is the added electrolytic solution. Furthermore, in tanks constructed of wood, even when lined with tarred felt and "asphalted," the penetration of the acid $CuSO_4$ cannot be prevented. Wooden tanks soon become efficient conductors and lose considerable current by short-circuits around the sides of the tanks, and leakage to the ground. The foremen and superintendents of series processes are only too well acquainted with such short-circuits and leakage. And the older the tanks, the more subject are they to these disorders. So that under the series arrangement, with new tanks, at highest, only about 90 per cent. efficiency can be maintained; *i.e.*, only 90 per cent. of the theoretical deposit is obtained for a given current; and with the lapse of time this keeps continually falling. A few years ago, daily examination in one of the best conducted series refineries in the United States showed an efficiency below 85 per cent.

There is another kind of short-circuiting possible, as when the anode makes contact with the opposite cathode surface. In series processes this affects only a single electrode, the remaining electrodes in the tank getting their full complement of copper. It is urged against the multiple system, that such short-circuiting affects, until discovered, the depositing capacity of the entire tank. This is true; but it does not constitute a valid argument against the system. In electrolytic practice, such short-circuits will occur and temporarily reduce the efficiency of the system. But the prevention of this lies with the foremen and superintendents, and can be reduced, by arranging the tanks in series, to a condition where short-circuiting would have to occur in at least three or four tanks before the efficiency was impaired by 1 per cent. With the low electromotive force required and the tendency of the current to flow through the electrodes and conductors instead of around the sides and bottom of the tank, as in series processes, an efficiency of 95 per cent. is possible under the multiple arrangement. This is a condition that has been attained and maintained in such refineries.

With regard to the copper held back in the two systems, the multiple has the advantage; for in two given plants having the same cathode surface and operated under the same conditions of density of current and refining constants, the multiple system requires only one-half the anode copper used in series processes. Where a refinery ties up an amount of copper, depending on the output, varying from 125 to 900 tons, the factor of interest on one-half of this is not to be disregarded. Of course, in series processes the anodes can be made thin, so as to obtain the maximum depositing surface with minimum weight of anodes. But the thinner the anodes the greater care must be exercised in preparing them, and the more frequently they have to be renewed; and the fixed charges entailed by these circumstances about balance the saving in interest effected by using the thinner electrodes.

As to the element of relative expense of operating under the two systems, the advantage is with the multiple. In order to compensate for the extra copper used in series processes, it is the rule to economize in tanks and solution by arranging the electrodes closer together than is done in the multiple system. With electrodes close together there is the possibility of sprouting and short-circuiting, unless the current is maintained of uniform density over the entire surface of the electrodes. Hence arises the necessity in series processes of working the anode copper up to a point where it is uniform, and following this by poling.

In the multiple system, the copper does not necessarily have to be improved and poled, although this procedure is common to many works where the multiple arangement is adopted. At some works, notably the Boston & Montana Consolidated Copper and

Silver Mining Company, the anodes are cast direct from the converters, thus saving the entire cost of remelting and refining the anode copper inherent in series processes. To neutralize the inevitable consequence of unequal corrosion following from such procedure, it is customary to place the electrodes from 2 to 2½ inches apart. There is not then the necessity for maintaining uniform density of current over the entire surface, this inequality of density in the rougher plates being overcome by changing the cathodes rather more frequently than the anodes. Short-circuiting from sprouting is thus prevented, and the inequality of density of current neutralized. In the series system, "stripping" the deposit cannot be economically practised until the tank, as a whole, is ready to be emptied.

Of course, the use of less pure copper in the anodes, in multiple arrangements, tends to make the electrolyte impure and increase the cost of refining by necessitating renewals of the electrolyte. This is only true, however, where no effort is made to keep the electrolyte free from the impurities that enter it from the anodes.

At first sight, it would appear as if bringing the electrodes closer together—say one-half the distance in the multiple arrangement—would, by reducing the resistance between electrodes, and making possible twice the output per horse-power mechanical energy expended, thereby justify the extra expense in preparing the anodes. While the output is increased in this way, it cannot at best be more than double the ouptut under a multiple arrangement, save where the anodes are rolled plates. Here, the resistance between electrodes is as low as one-third that between electrodes in a multiple tank. The expense, however, of rolling the plates, and the greater cost of stripping, indicate that economy cannot be effected along these lines. In a general way, it may be stated that the cost of improving and poling anode material exceeds the saving resulting from increased output per horse-power of mechanical energy expended. The multiple system, moreover, is free from the costly process of "stripping" the deposited copper from the anode scrap. In one series process this is frequently so difficult a matter that the deposited plate is often thrown back into the blister-furnace along with the rest of the scrap. Furthermore, the cost of maintenance of plant is greater in series processes, owing to the fact that the life of the tanks (when made of wood) is limited to about four years. Series arrangements gain slightly from the circumstance that there is less loss of energy in the conductors than in the multiple system, where there is always a consumption of about from 5 to 8 per cent. of the mechanical energy of the circuit. Where coal is cheap this is not important. The interest charges on plant are also slightly in favor of series processes. This favorable factor of the series is offset by heavier cost of renewals and larger interest on stock. The series system is free from the expense of making cathodes, inherent in the multiple system, but is still subject to heavy charges for "stripping," amounting to four times that of making cathodes. Balancing these various factors, as well as is possible in two works operating under the same conditions of cost of labor and fuel, there is a saving in operating expenses, in using the multiple system, of $1.98 per ton of refined copper. This difference is susceptible of greater increase, if the refinery is run as part of a converting establishment; for, since anodes may be made direct from the converters, the expense of making them in the reverberatory (amounting to $3.40 per ton) is altogether avoided.

The basis for the above figure is evident from the analysis of expenses of operating under the two systems. The subjoined figures represent the costs of electrolytic treatment of converter blocks, or other pig copper, but do not include the production of ingots from the cathodes. As the price charged for this is usually one quarter of a cent per pound (which leaves a small profit to the refinery), $5 added to the totals below will give the costs complete of making ingot copper from blister copper. If wire bars of high conductivity have to be made, the cost will be greater. The figures below represent cost per ton of cathode copper produced in a plant with a daily output of 30,000 pounds.

	Multiple.	Series.
Making anodes (includes reworking slag)	$3.40	$5.57
Fuel at $4 per ton (in electrolysis)	3.66	1.73
Consumption of energy in conductors	.26	.02
Expense involved by loss of efficiency	.18	.54
Treating slimes	.90	.90
Labor (including superintendence)	3.68	4.14
Interest on $82,000 and $62,000, respectively, at 5 per cent.	.74	.55
Interest on stock of copper in process	1.41	2.60
Maintenance of plant	.76	.97
Oil, waste, light, etc	.17	.12
Totals for refining, exclusive of insurance and office expenses.	$15.16	$17.14

An analysis of these figures indicates that the saving of \$1.98 per ton under the multiple system represents, approximately, the difference between improving, and not improving, the anode copper. Improving this material under the multiple system does not, however, wipe out this difference, there being still a saving of \$1.74 per ton; for, as the fuel consumption in making anodes increases, that factor in electrolysis is diminished, though not to an equal degree.

Regarding the first cost of construction, the simple unpatented series system has an advantage, as it does not come in for the element of tank-conductors, nor for the lead lining of the tanks, and is cheaper so far as the steam and power plant is concerned. As to the wooden part of the tanks, the cost of construction under the multiple arrangement is lower, and this partly offsets the extra expense of the lead lining. If much expense is gone to in making a series tank thoroughly solution-tight by means of tarred felt, tar, and asphalt (especially where asphalt is forced into the fiber of the wood under pressure), the cost may approximate that of the lead tanks of the multiple system. Where slate tanks are employed, the first cost, as stated, is greater. So that the difference in the cost of the two plants lies in the extra expense of the tank-conductors and plates for making cathodes, plus about one-half of the value of the lead lining of multiple tanks, plus one-third the value of steam and power plant.

In summing up this question, it can be said that it is generally conceded by electro-metallurgists, that the multiple system, in spite of its larger first cost, is susceptible of greater economy than is possible under series arrangements. Considerable weight is lent to this statement by the fact that, after costly and exhaustive experimentation with a series process, the Anaconda Mining Company finally discarded it and installed, under Mr. Thofehrn, the multiple system. The multiple system is now used also by the Baltimore Copper Smelting and Rolling Company, of Baltimore, Maryland; the Balbach Smelting and Refining Company, of Newark, New Jersey; the Chicago Copper Refining Company, of Blue Island, Illinois; the New England Electrolytic Copper Company, of Pawtucket, Rhode Island; the Boston & Montana Consolidated Copper and Silver Mining Company, Great Falls, Montana; Omaha and Grant Smelting and Refining Company, ~~Omaha, Nebraska~~; the St. Louis Smelting and Refining Company, Cheltenham, Missouri; and by the Pennsylvania Salt Company of

Natrona, Pennsylvania, with a total output of about 250,000 pounds daily. The same system is also to be established at the new works at Perth Amboy, and also at one of the two other new Eastern refineries predicted in the opening paragraph. The apparatus used in electrolytic operations will now be considered. To economize space, the discussion will be restricted to that used in the multiple system.

Electrolytic refining differs from most other branches of metallurgical work, in that a stoppage not only involves diminished output and increased cost of production, but may possibly be attended by a deterioration in the quality of the output itself. Hence, the boilers, engines, and dynamos must be selected with a view to running continuously and indefinitely, without even temporary shut-down.

The boiler plant should include reserve capacity that can be drawn upon whenever it becomes necessary to shut down that in actual use. Moreover, as fuel forms a considerable part of the expense of electrolytic refining, the boilers should of course be of a design to effect large production of dry steam with minimum use of fuel.

The same remarks regarding economy apply to the selection of an engine. It is, furthermore, necessary to have an engine that can be relied upon for absolutely continuous work. It should be simple in design and not possess too many moving parts. For this reason, although gas engines hold out promise as far as economy is concerned (the 320 horse-power gas-engine used at the Pantin Mills, Paris, consumes but 0.811 pounds of a rather inferior coal per independent horse-power, and 1.03 pounds per brake horse-power hour). Still, the great multiplicity of moving parts, and the lack of perfection in engines of large size, not to speak of the present cost—which is excessive—makes it doubtful whether gas engines can be used to advantage in electrolytic works. There is not so much choice with regard to the engine, provided it be economical and unusually free from liability to break-downs. The Corliss compound condensing engines constitute the best type for electrolytic work. Their economy is evident from the following table. Costs given do not include air-pump and jet-condensers.

Type.	Pounds Coal per Horse-power Hour.	Cost per Horse-power, sizes above 100 Horse-power.
Corliss single.	$3\frac{1}{4}$ to $3\frac{3}{4}$	$8.50 to $10.50
Corliss single condensing....	$2\frac{1}{2}$ to 3	
Corliss compound condensing	$1\frac{2}{3}$ to 2	13.00 to 15.00

The generators used in electrolytic refineries in the United States vary largely in design, according as they embrace the Edison, Thomson-Houston, General Electric, Mather, or Hochhausen types. The first three are made by the General Electric Company, of Schenectady, New York. The Mather dynamo is built by the Eddy Electric Manufacturing Company, of Windsor, Connecticut, and the Hochhausen dynamo by the Brush Electric Company, of Cleveland, Ohio. By far the largest number of depositing machines have been built by the General Electric Company, which has achieved great success in the various types produced at their works. Of the 42 dynamos known to the writer, that are being used for copper refining, 35 (with an aggregate capacity of 2,835,500 watts) were constructed by this company.

As the scope of this chapter is limited, a discussion of the value of different types of generators cannot be attempted. The capacity required being known, the design can safely be left to the manufacturers. It is in every case advisable to have the machine built by those companies only that have had long experience in this line of electrical construction. In this connection it might be said that dynamos are usually constructed under specifications that bind the manufacturers to produce machines of a certain electrical and commercial efficiency which, even after long running under a full load, will not heat very much above the temperature of the surrounding air. These conditions govern to a certain extent the design of the dynamo. The prices of different machines seem to agree very closely and range from $22.50 to $25 per K. W.

In ordering a generator, too much importance cannot be attached to the following points:

1. The mechanical construction should be of the most rigid and permanent character.

2. It should be of such design that it will run continuously without requiring any attention other than what can be given to it while in motion.

3. The bearings should be of large size, self-oiling, and self-aligning.

4. The speed of revolution should be low; not more than 300 to 350 revolutions per minute.

5. Large generators should be made multipolar, in order to divide the total current, so that the collecting device may be of convenient size and sufficiently light to insure good commutation.

6. Although no sparking is to be apprehended, the commutator segments should be made to allow a wearing of at least 2 inches, thus insuring indefinite life.

In order to enumerate the qualities a generator should have, a description can be given of some of the machines now under construction by the General Electric Company. These, according to a letter received from the power department of this company, have capacities of 1,500 amperes and 133 volts, or 1,333 amperes and 150 volts—and indeed are of such construction as to give 200 K. W., either at a high pressure and low current, or high current and low pressure equally well, without any material change in the rest of the machine. The speed of such machines is 300 revolutions. Other dynamos, also under construction at the present writing, have a current of 1,500 amperes and 120 volts, and run at 240 revolutions per minute. The armature of these machines is of the "smooth core type," with Siemens drum winding. The windings are all the same and interchangeable, so that in case any portion becomes injured, the winding can be replaced with but little difficulty. The commutator is built on a separate spider from the armature proper, and can be easily removed and replaced. The commutator shell is divided into sections, so that individual segments can be removed without taking commutator from the armature shaft, and without disturbing the adjacent segments. The low voltage between adjacent segments, the slow speed, and the balance of the various parts, combine to make a generator that will run without sparking, and allow the voltage to be varied by changing the strength of the field. The field magnets are supplied with rheostats. The machine is designed to run continuously, under full load, with 92 per cent. commercial efficiency, 95 per cent. electrical efficiency, and guaranteed not to heat higher than 40 degrees C. above the room. Fig. 77 represents a 200 K. W. machine manufactured by the General Electric Company.

Connection between the dynamos and tanks is made of high-conductivity copper. The shape of these conductors varies in different refineries, being round, square, or rectangular, in cross section. The theoretical size should be such that the interest on cost of conductor should equal the money value of loss of energy in conductor, due to its resistance. A conductor of at least one square inch cross section has been recommended for a current of 320 amperes. In practice, a much smaller conductor is taken, it being usual to allow one square inch for a current up to 666 amperes. This is somewhat small. In estimating the cost of an electrolytic refinery in the subsequent pages of this chapter, a circuit will be contemplated carrying 1,632 amperes through tank conductors of 3 square inch sectional area, and cross conductors ½

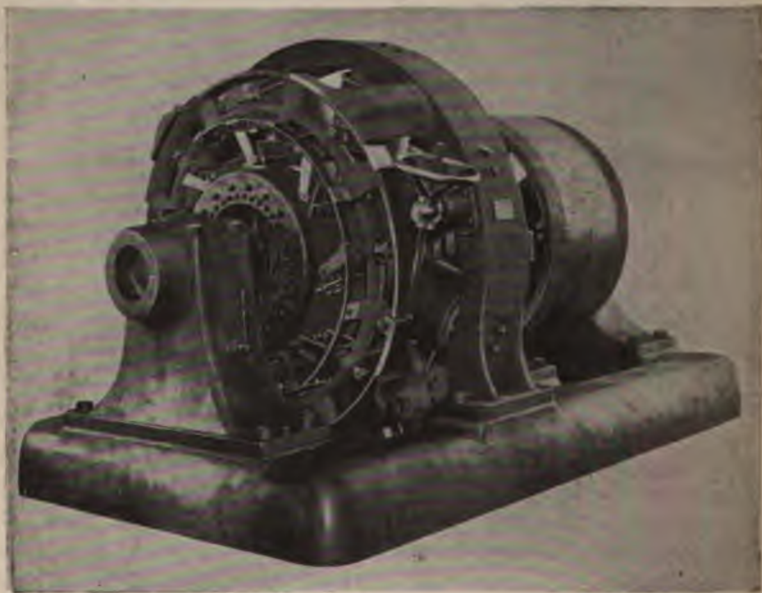


FIG. 77.—200 KILO WATT GENERATOR.

inch in diameter. The resistance introduced by these consumes 7.2 per cent. of the mechanical energy of the circuit. Rectangular conductors of 3 square inch sectional area weigh 11.67 pounds per running foot and cost 4 cents, plus the price of Lake copper, per pound. Round conductors ½ inch in diameter weigh 0.756 pounds per running foot, and cost 3 cents, plus the price of Lake copper, per pound.

The anodes used contain from 98 to 99 per cent. copper. The most desirable anode material is that which is free from deleterious elements, such as bismuth, antimony, and arsenic. Presence of lead is not harmful, while the presence of a little tin is considered advantageous, as it tends to maintain the purity of the electrolyte.

The dimensions of anodes vary in different works. The size must depend upon ease of handling into tanks, of cleaning surface of contact with the conductor, and non-liability to short circuits. As large anodes do not make as much scrap proportionately as small ones, their use, when length is greater than breadth, effects a saving in floor space and in tanks, as well as labor. Experience has suggested the use of anodes of from 5 to 7 square feet. Six square feet represents a good size. They are usually made $\frac{3}{4}$ inch thick. An anode 2 by 3 by $\frac{3}{4}$ inch weighs about 200 pounds and can easily be handled by one man. It is usually cast with two short arms, which rest on the tank conductors, being in direct

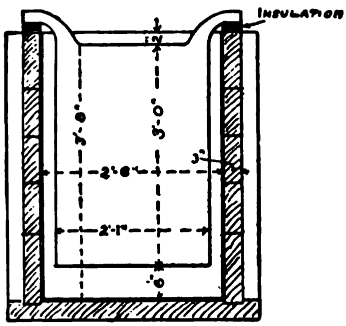


Fig. 78.—METHOD OF SUPPORTING ANODES IN TANKS.

contact on one side, and insulated on the other. In some works they rest on the bottom of the tank. This practice is strongly to be deprecated, as it produces short-circuiting through the slimes, and impairs in great degree the efficiency of the system. The manner of suspending anode is shown in Fig. 78.

The cathodes are made either in the regular refinery tanks or in special ones. When made in the former, using the two end electrodes in each tank as surfaces on which to deposit the cathodes, the cathodes will be of the same size as the anodes, and will have to be connected with the cross-conductors, either by straps riveted to the cathodes, or by means of clips. This involves an expense that can be avoided by making the cathodes about 8 inches longer than the anodes, in a separate tank that is deep enough to allow this. The plates on which the cathodes are deposited are sheets of rolled copper, say 3 feet 8 inches long, 2 feet 1 inch wide and

$\frac{1}{32}$ inch thick. These plates are prepared in the usual way, by slightly oiling, coating with good conducting graphite, and dipping the edges in melted pitch. They can be connected to the cross conductors by straps riveted to the plates.

With a current of 8.16 amperes per square foot, copper will be deposited a little faster than at the rate of 0.01 inch per day. As these cathodes are taken off when they get to be $\frac{1}{25}$ of an inch thick, it takes about four days to make one. In a series of 300 tanks, each containing 200 square feet of depositing surface, 30 tanks would be set aside for making cathodes. These would give 80,000 square feet of active cathode surface a month, which would allow the changing of the cathodes in the 270 refinery tanks about every 20 days. The advantage of changing the cathodes more

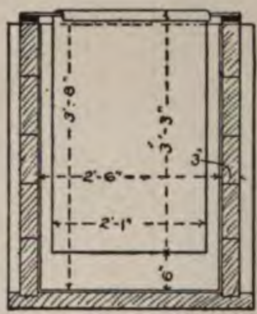


Fig. 79.—METHOD OF SUPPORTING CATHODES IN TANKS.

often than the anodes has already been pointed out. The manner of suspending cathodes is shown in Fig. 79.

Coming now to the tanks, consider first their dimensions. Large tanks, working under a current of low potential, are open to the objection that, if short-circuiting occurs, or an increase of resistance arises, the efficiency of the system is impaired to a greater extent than would be the case where there were numerous small tanks working under a current of higher electromotive force. Large tanks are also objectionable on account of the trouble of handling large electrodes. Too many small tanks, while increasing the efficiency of the system, make first cost excessive.

Expediency suggests that, in order to economize floor space, the tanks be made narrow and deep. Narrow tanks, moreover, are particularly desirable, as the workman has to lean across and arrange opposite side of electrodes. Length is determined by amount of cathode surface per tank and distance apart of electrodes. If the former is large, the density of current will have to be large to maintain an economical rate of deposition, and extra expense will be incurred in increased size of conductors and in

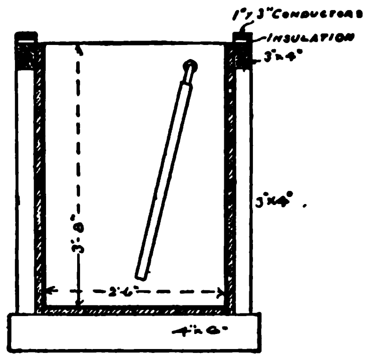


Fig. 80—Cross Section of Tank, Showing Tube and Base.

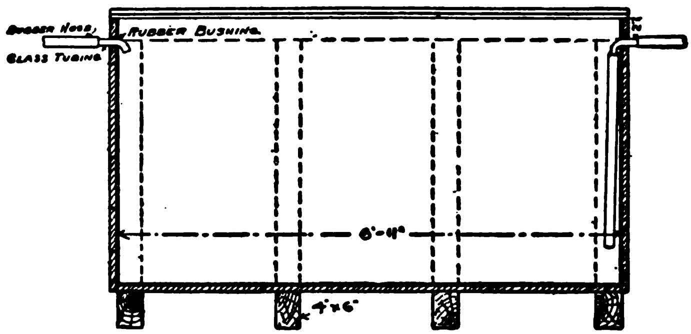


Fig. 81.—Longitudinal Section of Tank.

maintaining current through them. Experience suggests a mean of about 200 square foot per tank. With electrodes 2 inches apart, 3 and 4 inches, respectively, from each end of the tank, the electrodes of 6¼ square feet area, the tank would contain 17 cathodes and 16 anodes; and assuming the anodes are ¾ inch thick, the tank would be 83 inches long, 44 inches deep and 30 inches wide. This allows about 2½ inches between the edges of the electrodes and the sides of the tank, 2 inches from top of electrode to top of tank, and 6 inches from bottom of electrode to bottom of tank.

Owing to the limited space at the writer's disposal, a description of all the tanks used in electrolytic practice cannot be attempted. The simplest form of lead-lined tank and its connections will, therefore, alone be considered. According as the electrodes rest on the sides or on the frame of the tank, two different styles are possible. The first is the more expensive, but the more common, and consists ordinarily of 3-inch plank, bolted at sides and ends through the bottom, the bolts passing through the plan. In the other, the sides and end of tank are made of inch stuff, the bottom of 2-inch stuff; and being simply nailed to the frame, which consists of 3 by 4 inch scantling. The first style is shown in Figs. 78 and 79; the second in Figs. 80 and 81.

The tanks are lined with 6-pound lead. The use of lighter lead is not to be advised. These tanks are terraced, so as to have circulation going on through gravity, as this obviously saves a great deal of pumping. About 2 inches below the top of tank, to one side of center at both ends, a hole 1¾ inches in diameter is bored. The lead lining is made continuous through this hole to the outside of tank. Insert, until the ends are flush with inside and outside of hole, a piece of 1½ inch rubber hose. Into this, run at one end a piece of 1-inch heavy glass tubing, bent at right angles; and at the other end, a piece bent at 45 degrees. Connect the 90 degree bend with a piece of tubing reaching to about 6 feet from bottom of tank. By these means the electrolyte can be drawn from, and delivered to any point in the tank desirable. Connect the ends between tanks with a short length of rubber hose. This constitutes a form of circulation apparatus that gives a tight joint, and reduces to a minimum short-circuiting from tank to tank. The tanks rest on sills that have been coated with a mixture of tar and asphalt. The sills rest on piers of brick rolled in tar, and built up from a substantial asphalt floor.

The relative cost of the two styles of tank just described is given in the following figures. The totals show a saving of $4.18 per tank in favor of the style in which the weight of the anodes and cathodes is borne by the frame surrounding the tank.

	I.	II.
Lumber for tank (306 and 180 feet timber, respectively)....	\$3.06	\$1.80
Carpenter work..........	3.75	2.25
Lead, 95 square feet of No. 6, 570 pounds, at 3½ cents......	19.85	19.65
Lead burning..........	.90	.90
Iron bolts, 49 pounds, at 2½ cents..........	1.00	.....
Glass tube for circulation..........	.15	.15
Rubber tube for circulation..........	.92	.92
Paint, ½ gallon "P. and B." acid proof..........	.60	.60
Painting..........	.15	.15
Total for tanks proper..........	\$30.38	\$26.38
Add for stringers beneath tank..........	.82	.64
Add for brick piers..........	.90	.90
	\$32.10	\$27.92

The tanks are usually arranged in double rows, and between each of the two rows is a passageway 3½ to 4 feet wide, containing a narrow-gauge track. Underneath the floor, in each aisle, should be a lead-lined launder, connecting with a storage tank, which is itself in connection with the circulating apparatus. The tanks are siphoned into this launder whenever it is desirable to empty them in order to clean up the slimes.

Above each tank is an overhead crawl, running on a double track, and having on its under side a snatch block and fall. By the aid of this, one man can load and unload tanks very easily. Although claims are made in certain patent specifications that circulation of the electrolyte is not necessary, it is neverthe ess true that circulation *is* necessary, and is found beneficial in all cases. The amount of circulation will have to depend on the constants of the refinery. An average amount per tank would be that flowing through a pipe under a head of two inches, which represents the difference in level of adjoining tanks in the same circulation system. The means of circulation are numerous, and include plunger pumps worked from shaft of engine-room, plunger pumps run by small electric motor, injectors, and air pressure. Plunger pumps get out of order so easily that their use is not attended with any satisfaction. Injectors mix too much water with the electrolyte and cannot be depended on, unless watched closely. The simplest and best method is to have two small sheet-iron tanks, lined with lead, the covers also being lined and movable. A small air compressor (6 by 5½ by 5 inches) of the type given in Figure 82, manufactured by the Laidlaw-Dunn-Gordon

Company, of Hamilton, Ohio, is the source of the air pressure. It is connected up with a small air receiver, which, in turn, is connected with the two pressure tanks referred to. By forcing the solution out of one of these while the other is filling, the operation goes along without requiring any supervision whatever. Although numerous means can be employed for directing the air alternately from one vessel to the other, a very simple method was suggested

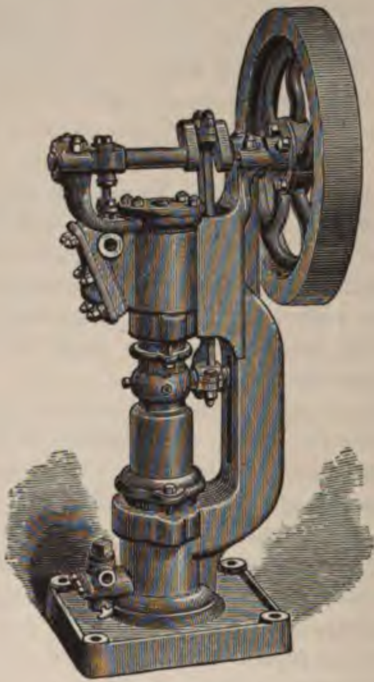


FIG. 82.—AIR COMPRESSOR.

to the writer by the late superintendent of the Anaconda Mining Company, Mr. Victor Ray. Reference to Fig. 83 will make this clear.

A is the solution tank, B and C the pressure tanks, D and E pipes to the launders that carry the electrolyte to the tanks, F the valve seat, G H, a rigid system revolved by gearing. It will easily be seen that, during half a revolution, air is being forced into one tank, and during the second half of the revolution into the other. According to the size of refinery and the amount of electrolyte to be circulated, the gearing and piping can be calcu-

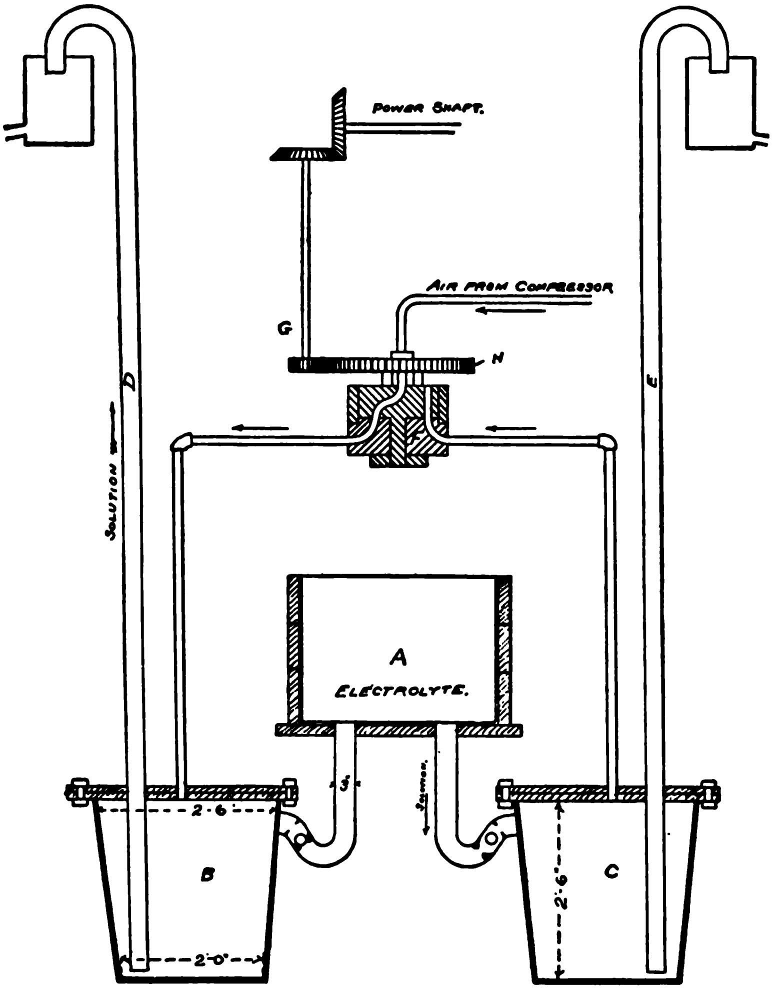


FIG. 88.—TANKS AND AIR VALVE.

lated so as to maintain any volume of electrolyte in circulation that is desirable. Mr. H. L. Bridgman, superintendent of the works of the Chicago Copper Refining Company, gets along with one pressure tank, by using a three-way valve, working automatically, which increases and relieves the pressure in this tank every thirty seconds.

Although, in a general way, a description has been given regarding the motive power, the electric generator, the tanks, etc., no mention has thus far been made of the size of engine, dynamo, and tank-room required for a given daily output. Before these questions can be settled, it is necessary to know the conditions under which the refinery is to be operated. As affecting the entire proposition consider first the rate of deposition.

The purity of the copper offered, and the use to which it is to be put, will determine the rate at which it can be electrolyzed. If it be reasonably free from antimony, bismuth, and arsenic, a higher rate of deposition will be permissible than where the opposite is the case. This rate varies at every establishment in the country, depending upon the above circumstances and the comparative expense of depositing at different rates. With anodes made from good converter blocks, a density of current of 8 to 9 amperes per square foot of cathode surface will be found a fair rate of running, and one at which the cathodes produced will make into conductivity copper. If the copper is free from deleterious constituents, a greater density of current can be maintained. Seven to ten amperes per square foot of cathode surface corresponds to the density of current adopted in electrolytic practice.

In speaking of density of current at cathode, actual density is meant, and not the density that is calculated from the reading of the ammeter. As stated before, the efficiency of different electrolytic systems varies considerably, the "multiple" giving a higher efficiency than the "series." Theoretically, a current of 10 amperes ought to deposit 10.032 ounces of copper per square foot of cathode surface every 24 hours. This theoretical output is never reached in practice. The highest recorded result represents an output of 9.84 ounces, corresponding to an efficiency of 98.08 per cent., a result that has been reached abroad. So far as can be ascertained, the highest efficiency that has been maintained in this country is one of 96 per cent. In calculating, therefore, the amount of current necessary to maintain a given output, allowance will have to be made to cover the loss of efficiency of the system.

Thus, if it be necessary to deposit 8 ounces per square foot of cathode surface daily, under a 95 per cent. efficiency, a current density of 8.16 amperes per square foot must be used, although theoretically 7.752 amperes per square foot would produce this amount, if the 5 per cent. loss did not enter the calculation.

With these data, it is easy to calculate the dimensions a refinery must have to maintain a given output. If the output is to be 30,000 pounds daily, and the rate of deposit 8 ounces per square foot, a total cathode surface of 60,000 square feet will be necessary. If it be decided to divide this into 300 tanks, each tank will have 200 square feet of depositing surface. The floor space required for the tank-room would be about 150 by 120 feet. The amount of copper in process, 570 tons. The tanks under such an arrangement would be about 83 inches long, 30 inches wide and 44 inches deep, all inside measurements.

The electrodes would number 33 per tank, of which 17 would be cathodes and 16 anodes. The area of the electrodes would be 6.25 square feet on one side, the end cathodes being supposed to receive the deposit on one side only. If the 300 tanks are in series, then the total amperage required will be 200 times 8.16, or 1,632 amperes. If the voltage were known, the size of dynamos and engine would also be known. To ascertain them, let us consider the question of voltage.

The determination of the electromotive force necessary to maintain a current of a given intensity in a circuit of known resistance, involves a very simple calculation. The determination of the electromotive force necessary to maintain a current of given density in an electrolytic refinery involves a very complex calculation. Indeed, an absolute figure cannot be ascertained; for the resistance of a refinery circuit is made up of the internal resistance of the dynamo, the conductors, the electrodes, the electrolyte, and other factors, and these are constantly changing; frequently reinforcing one another to make the combined resistance higher; occasionally combining, under short-circuiting, to make the total resistance lower. So that it is not unusual for the manager about to construct a refinery to take a constant, from his own experience, of voltage per tank, and base the total voltage upon the number of tanks in series. Then with a dynamo of given capacity, the plates would be suspended in the vats until a certain voltage had been used up, when the next tank would be charged in precisely the same way. As a case in point at Anaconda, where the series system was in use, an allowance of 2 to 10 volts between plates was allowed. As each tank had 60 plates, an electromotive force of 12 volts was required per tank. In charging a tank, the plates No.

were put in until a voltmeter indicated a fall of potential of 12 volts at the terminals of the tank; whereupon, that tank would be considered charged. According as the plates are in multiple or series, an allowance is usually made of $\frac{1}{8}$ to 12 volts per tank.

The theoretical calculation, if it is desired to make one, would be as follows: Assume a series of 300 tanks with 200 square feet of cathode surface, joined in parallel in each, and electrodes 2 inches apart. As mixed solutions of copper sulphate and sulphuric acid conduct somewhat better than the mean of the constituents, the resistance of the ordinary electrolyte, at 75 degrees F., is about 14.5 ohms per cubic centimeter of solution. For a tank containing 200 square feet of cathode surface and electrodes 2 inches apart, the resistance of electrolyte per tank would be 0.000396 ohms; and for 300 tanks, would be 0.1188 ohms. For the internal resistance of the dynamo, no absolute figure can be given, for this element varies in different generators, and even in the same generator, between small limits, all the time the dynamo is running. For a dynamo of 1,632 amperes and 215 volts capacity, an internal resistance of 0.002 ohm can be assumed. For a given temperature, the resistance of conductors can be determined easily, when the length, cross section, and specific conductivity are known. The resistance interposed by the conductors of a refinery of 300 tanks of the size given, the conductors being at least 3 square inches in cross section, and the cross conductors of $\frac{1}{2}$-inch rolled copper, is 0.0095 ohms, corresponding to a consumption of about 7.22 per cent. of the total mechanical energy employed in the circuit. The remaining resistance is a complex of many factors, made up of obstruction at anodes from adherence of insoluble coating, and of various secondary actions taking place in the tanks themselves. This resistance is very variable, but may be put down at 0.0013 ohms. As the total resistance of the circuit must equal the sum of the separate resistance, we have (in ohms):

$$R. = 0.1188 + 0.002 + 0.0095 + 0.0013 = 0.1316 \text{ ohms.}$$

$$\text{When, since } E = R., C = 1{,}632 \times 0.1316 = 215 \text{ volts,}$$

Therefore, to deposit 30,000 pounds copper in a refinery of 300 tanks, under the conditions just mentioned, requires a generator of 1,632 amperes and 215 volts. This corresponds to a production of 2.4 pounds, per horse-power hour, of mechanical energy employed. In practice, it is usual to select dynamos having a somewhat greater capacity than needed, in order to be able to run the generator at a slower speed than specifications call for, and at the same time to have additional capacity, if needed. The capacity of the engine is calculated by multiplying the volts and amperes together and dividing by 746. This will give the electrical horsepower required, upon which a sufficient margin will have to be made to cover losses in the dynamo and engine, and allow for any extra demand that may occasionally be made upon it.

In starting up a refinery, the cathodes will first have to be made. To charge the thirty tanks in which the cathodes are to be deposited will take about two days.

On the fifth, and each successive day, the cathodes that are made can be suspended in the regular refinery tanks. As fast as extra tanks are introduced into the circuit, the voltage of the current is increased, either by diminishing the field resistance or by increasing the speed of the dynamo. Ordinarily, the former method is adopted. Under this arrangement, the generator is started at zero voltage, and the voltage increased as needed, by means of the rheostat connected with the field. When all the tanks are filled, the ammeter and voltmeter will indicate, respectively, the quantity of current passing through the circuit, and the fall in potential at the terminals of the machine. By a simple process of examination, it can be ascertained how much of this current is being exerted in the act of electrolysis.

It would seem superfluous to urge upon those interested the necessity of the closest supervision over the operations in the refinery, were it not for the fact that a sense of false economy has often led the manager to be satisfied with the most superficial examination. The writer knows of instances in which the resistance per tank became so great, in a certain series-refinery, as to cause heating to a point at which the paraffine melted from the electrodes and floated to the surface. Superficial examination of the circuit would be all right were the only phenomena to be observed those of heating, caused by improper contacts, darkening of the cathode (which follows too rapid deposition of copper), evolution of gas from too great density of current, and similar phenomena. But such disorders are unusual in good refinery practice, and this suggests the necessity of some better means of examination than the merely superficial ones. In the brief scope of the present article, only the most important irregularities connected with electrolytic practice can be touched upon; but these will be found to require the utmost attention, if high efficiency and high-grade of product are to be maintained.

To determine whether the current is doing its full work, recourse can be had to the simple expedient of weighing any one of the cathodes in each of the tanks. As 8.16 amperes will deposit 8 ounces of copper per square foot of cathode surface daily, under a system efficiency of 95 per cent., the weight of the plates will indicate what efficiency is being maintained. Ordinarily, most of the tanks will be found to give their full complement of copper. If, in one or two or three tanks there be found a smaller deposition than is expected, the trouble will be found, in all probability, to consist in a short-circuit, caused either by sprouting from the sides of the tank, from the cathode to the opposite anode, by a short-circuit through the slimes, or lastly, by a falling over of some of the anode scrap against the cathode. Any short-circuit like this, in the multiple system, impairs the efficiency of the whole tank, and must be carefully guarded against. To ascertain whether the tank is short-circuited in the method described, recourse can be had to a voltmeter, which is connected with the terminals of the sus ected tank, and the reading made. A constant fall of potential should be observed in all the tanks. If any tank shows a smaller fall of potential than would be anticipated from the regular constants of the refinery, it is an indication of short-circuiting. A rise of potential, on the other hand, will indicate an increase of resistance, such as might be caused by adherence of insoluble coating on the anodes. When this is observed, the anodes must be cleaned. In a generator designed to produce a constant current under a slightly varying resistance, this circumstance will not show itself by diminished output, although its effect will be seen by greater consumption of mechanical energy. Any leakage to the ground can be detected by an instrument used by electric-light men in locating and determining the amount of ground circuits. This consists of a set of incandescent lamps arranged in parallel on a baseboard, and governed by a switch. When the ground current is of sufficient intensity, one or more of the lamps will light up, depending upon the strength of the current and the amperage at which the lamps will burn.

In most works, a large ammeter and voltmeter, in the dynamo room, register the total current and total fall of potential in the circuit. It is necessary, however, to examine the fall of the potential through the entire circuit, in order to guard against short-circuiting in any individual tank. Portable voltmeters are handy for this purpose, and can be had from any of the electrical instrument makers. The "Keystone" and the "Weston" voltmeters are especially to be recommended.. A sketch of a Weston direct-reading voltmeter is given in Fig. 84.

This is the style suitable for examination of the current throughout the circuit. The divisions in the upper part of scale

FIG. 84.—VOLTMETER.

represent single volts, and are readable to tenths, while the divisions in the lower part of scale are tenths of a volt, readable to hundredths.

Besides the testing of the current, a thorough examination will have to be made of the anodes, the electrolyte, and the cathodes. Of the anodes, the complete analysis is not absolutely indispensable, save as affording means of tracing the cause of any increase of impurity in the electrolyte, and, consequently, in the cathode copper. In order not to crowd the laboratory too much, the analysis of the cathodes could be left until any fall in the conductivity of the output might raise the question as to the constitution of the copper of that particular lot. Again, as the laboratory will have ample work in assaying the blister copper, and in making conductivity tests of the output daily, the thorough analysis of the electrolyte need only be made say twice a week. It is advisable, however, to make rapid determinations of copper and sulphuric acid (free and combined) daily, as a check upon the operations in the refinery.

It is generally believed now that there are few "secrets" in electrolytic practice, and that the production of a high-grade of copper is attainable by any refinery that will practice deposition at reasonably slow rates, and maintain the most thoroughgoing supervision over all operations. As all electrolytic work is a balance, or equilibrium, of conditions, it is necessary to maintain all conditions uniform. Hence, besides the analytical tests on the electrolyte, it is advisable to keep a record of daily tests made with thermometer and hydrometer, and maintain the electrolyte constant with respect to its temperature and its condition.

The composition of the electrolyte does not, apparently, vary largely in different refineries. It is usually made up in the following proportions by weight: Water, 75 parts; bluestone, 19 parts; and 66 degree Beaumé sulphuric acid, 6 parts. The limits appear to be from 15 to 20 per cent. bluestone and 5 to 6 per cent. acid. In making up the electrolyte, a tank of known cubic capacity can be filled up to a mark. As water weighs 62½ pounds per cubic foot, the weight is easily ascertained. Add 15 to 18 per cent. bluestone, and, after it is dissolved, acid until a Beaumé hydrometer indicates 16 degrees to 18 degrees, corresponding approximately to a specific gravity of 1.12 to 1.15, with water taken as unity. Ordinarily, the electrolyte has a tendency to maintain itself reasonably pure by the reactions that take place between the impurities of the anode copper. Thus, the arsenic is partly precipitated by union with the lead of the anode, after the former has become an acid radicle. Arsenic also combines with tin, acting in this case as a base, the tin oxide constituting the acid radicle. Bismuth and antimony are slowly converted into oxy-compounds by the influence of the air. This suggests two means of maintaining the purity of the electrolyte. The first comprehends the addition of a small percentage of tin to the anode copper, and the second, the oxidation of the bismuth and antimony by jets of air forced into "the collecting basin and distributing reservoir." The value of this latter method was demonstrated by Mr. H. Thofehrn, the present superintendent of the electrolytic department of the Anaconda Mining Company. If the solution becomes too impure from presence of iron salts, which consume energy in their reduction to proto-salts at the cathodes, recourse must be had to recrystallization of the electrolyte, or else, deposition of the copper contained by electrolytic methods, using insoluble anodes.

The handling of the copper in and out of tanks is very simple. The copper is brought into the refinery on a four-wheeled truck, running on tracks in the passageway between each two double row of tanks. Two men can bring 30,000 pounds in, and arrange them on end, at the ends of the tanks, in three hours. Under the arrangement of a straight track running down the center of a whole row of tanks, it is necessary, where the circulation pipes are in the center of the tanks, to draw the anodes up by means of the block and fall attached to the overhead crawl, and push the crawl from end of row up to the tank where the copper is to be charged, passing back again to the end of the row for the next anode. This obviously necessitates a great deal of passing to and fro, which might be avoided by having the circulation apparatus at end of each tank placed at one side of center. The bends at end of glass tubing allow solution to be drawn from, and delivered to, any part of tank desired. The electrodes are thus raised through space between ends of tanks, and no unnecessary handling caused by working the crawl over more than one tank. The use of a narrow tank, only 30 inches in width, and electrodes that do not weigh over 200 pounds, enables two men to raise and place nearly 30,000 pounds of anodes in a single day of ten hours.

It was stated in a previous paragraph, that for the double purpose of maintaining the density of current uniform over the cathodes, and of not having too great a weight on the cathodes (which are supported by a bar of ½-inch copper), the cathodes are changed more frequently than the anodes. If the anodes are changed say once a month, then it is advisable to change the cathodes every twenty days. Hence, every day the cathodes from fifteen tanks would be removed and sent to the refinery. Two men, working eight hours apiece, can unload the cathodes and insert new ones in fifteen tanks. The cathode copper is removed by the same block and fall with which the anodes were handled, loaded on trucks, and sent to the refining furnace.

The method of preparing the cathodes is well known, and needs no detailed explanation. As stated before, they are precipitated on sheets of rolled copper about $\frac{1}{32}$ inch thick—having previously been slightly oiled, and coated with good conducting plumbago. In four days the cathodes are ready. In order to make them hang vertically, they must be bent so that the center of gravity comes exactly under the cross conductor on which they are suspended.

At the end of one month, the tanks first charged may have collected about 20 pounds of slimes each, containing in ten tanks perhaps 1,350 troy ounces of silver and gold. The conditions prevailing at the refinery will determine how often the slimes are to be worked up. If it be desired at the end of one month to begin treating 1,350 ounces daily, the solution will be siphoned out of ten tanks into the launders that connect with the circulating system. It is not considered advisable to wash off the electrodes until the greater part of the slimes have been removed from the tank. The anodes are then scraped and washed with water, and the slimes removed as before. Although the slimes can be handled cheaply by means of a suction pump, the danger of loss of slimes by "blowing" makes it desirable to dip out the slimes and carry them in a barrel standing on a four-wheeled truck to the slime tank. A piece of rubber hose is connected up to the lead nipple in the bottom of the barrel, and closed with a large pinchcock. At the slime tank, the slimes are run on to a screen of about 16 meshes to the linear inch, and the silver washed through with the electrolytic solution. The copper scrap, of which there is a great deal, is sent back to the anode furnace. When the slimes have settled, the supernatant solution is returned to the regular circulation system. The second lot of slimes from the washed plates is then added to the first, and the two stirred up with addition of water. This supernatant solution can be saved for making new electrolyte. Other wash water, that is more dilute still, can be run into tanks containing wrought iron, and the copper precipitated, or the copper can be deposited by the current in separate tanks, using insoluble anodes. The slimes can be thrown on to a filter box containing a false bottom, and drained. They are then ready for the refining operation.

Although the composition of these slimes varies between wide limits, according to the care taken in washing and screening, with the object of diminishing the copper contents, an average composition would be as follows:

Silver 50 per cent., gold 0.5 per cent., copper 15 per cent.—the remaining 34.5 per cent. consisting of valueless impurities.

Of the various processes adopted for treating these slimes only one need be considered, that being the one employed at Anaconda, the details of which were worked out, we believe, by Mr. Ray. The process, in point of view of costs of operation, is the most economical in use, and deserves a full description. The advantage (or value) of this process is, that on days when the base bullion is being "cut-up" by sulphuric acid, the fume of SO_2 and H_2SO_4 are used to dissolve the copper contained in the slimes, without the addition of other acid.

On days when the regular parting is not going on, the slimes are put into a lead-lined tank with about an equal weight of a mixture of three parts of 66 degree acid and one part water. Steam and air are then introduced through a perforated lead pipe, connected up with a Koerting injector. Through the action of the H_2SO_4, air, and steam, the copper is entirely dissolved from the slimes. On days when parting is going on, and considerable SO_2 is being liberated, together with fumes of H_2SO_4, these gases are introduced through the same apparatus, together with the steam and air, into the tank containing the slimes, which, in this case, are well covered by the ordinary electrolyte. The reactions that take place are those of the Roessler converter; that is to say, the SO_2 reduces the cupric sulphate to cuprous sulphate, while the air oxidizes it back again to cupric sulphate. In the simultaneous reduction and oxidation that goes on, free sulphuric acid is formed, and this is in part neutralized by the copper contained in the slimes. No heating is necessary, beyond what is furnished by the steam and the reactions taking place. When the copper is dissolved, the slimes are thrown on a filter, washed, dried, mixed with one-fifth their weight of soda-ash, and smelted on the hearth of a small reverberatory furnace, to a base bullion, about 980 fine. A more economical furnace would be one similar to an ordinary English cupel furnace, the hearth of which would be supported by iron stringers connected up with a right and left-handed screw-mechanism for raising and lowering hearths.

The method of parting silver and gold is so well known that a detailed description is not necessary. The silver is dissolved in an iron kettle, shown in Fig. 85. The silver-sulphate formed is then drawn into a lead-lined tank and reduced with metallic copper, ferrous sulphate, or metallic iron. If iron is used, care must be exercised that no copper is precipitated. If copper is used as a reducing agent, the copper sulphate formed is crystallized and used in the maintenance of the electrolyte. The acid mother liquor can be used for dissolving copper out of new batches of slimes. The precipitated silver is washed, dried, and melted with a little nitre in plumbago crucibles. The cost of this refining of

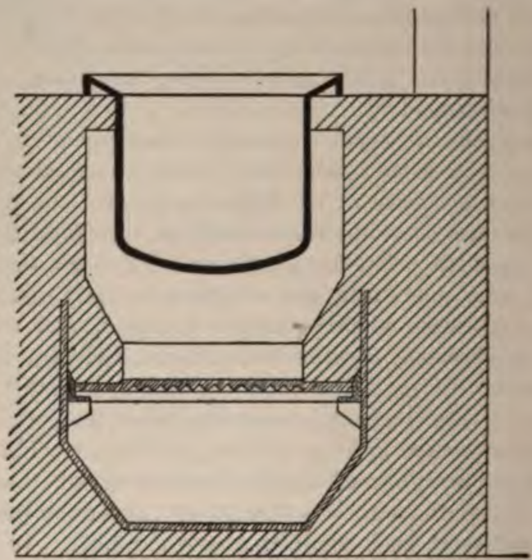


FIG. 85.—PARTING-KETTLE.

the slimes is given below, the estimate being based on the production of 1,350 ounces of silver and gold daily. In a plant producing 30,000 pounds of copper daily, the cost of treating the slimes adds about $0.90 per ton to the cost of the electrolytic refining of the copper, as shown by the following summary:

Labor	$9.26
Coal	1.72
Acid	1.34
Nitre and soda	0.30
Deterioration of plant	0.83
Total	$13.45

Or about one cent per ounce of silver and gold contained in the slimes.

While this process is exceedingly economical, so far as cost of operating is concerned, it is open to the objection that the loss of silver in smelting the copper-free slimes to a base bullion must be rather large. Just what the loss is, cannot be stated, as there appear to be no data obtainable bearing on this subject. It seems inevitable, however, that smelting so much silver, combined with volatile compounds of lead, antimony, and arsenic, must occasion serious losses. For this reason, it appears to the writer that it might be more economical to treat the moist slimes with a mixture of two parts 66-degree acid and one part water, at a temperature of 140 degrees C., making up the water lost by evaporation, until the copper is all dissolved. The heat is then increased until all the free acid has concentrated and combined with the silver. Enough of the acid is added to dissolve all the copper and 75 per cent. of the silver. The sulphate would then be leached out by hot water and precipitated by metallic copper. The copper sulphate formed is concentrated, crystallized, and used in maintaining the electrolyte. The mother liquor can be used in making up dilute acid for subsequent operations. If the remaining slimes are now melted to a base bullion, the loss will be less, as there is only one-fourth the amount of silver present that there is in the other method. Moreover, this silver contains proportionately more gold, and there is therefore less chance of losing gold in parting, since it is well known that the greater the amount of the gold, within certain limits, the better it holds together, and the less likely is it to be siphoned over with the silver sulphate, when the latter is sent to the reduction tank.

It was the writer's intention, in preparing this chapter, to conclude with a detailed statement of the cost of building a refinery. Owing to the large amount of space necessary for such a statement, his intention must be abandoned. As a substitute, a condensed statement will be given of cost of different parts of an electrolytic establishment operating under the multiple system, and with an output of 30,000 pounds daily. The cost, as will be seen from the total, is in the neighborhood of $82,000.

Buildings covering area of 25,450 square feet, requiring 500,000 feet lumber, at $9	$4,500
Framing 500,000 feet lumber, including windows, nails, pins, spikes etc., at $16	8,000
1 reverberatory furnace (10 foot by 16 foot hearth), with anode molds	4,000
1 reverberatory furnace (10 foot by 16 foot hearth), with water bosh, and ingot, and wire-bar molds	4,500
1 36-inch water-jacket blast-furnace, for reducing slag, complete with electric motor, fan blower, and 12 slag-pots	1,200
Excavations for boiler, engine, and dynamo-foundation, 1,200 cubic yards, at $0.25 a yard	300
Foundations, 10,000 cubic feet, at $0.16 (includes stone, cement, and mason-work)	1,600
Five 72 inch by 16-foot R. S. boilers, including brick-work, at $16 per horse-power	8,000
Air pump and jet condensers	1,390
Two 300 horse-power compound condensing Corliss engines at $18 per horse-power	7,800
Two 180 K. W. direct current, shunt-wound generators, at $23 per K. W.	8,280
700 yards asphalt flooring at $0.75	525
Circulating apparatus, including air pump	400
Receiving tanks, distributing reservoirs, and launders	245
3,500 feet 1-inch pipe, 100 1-inch tees, 100 1-inch faucets	220
300 lead-lined tanks, with stringers and brick piers, at $27.92	8,376
Overhead track	600
Silver refinery	3,500
Rectangular conductors, 58,350 pounds, at 14c	8,169
Round conductors, 11,566 pounds, at 13c	1,504
Rolled plates for making cathodes, 3,060 square feet $\frac{7}{32}$-inch thick — 31,212 pounds, at 13c	4,057
Contingencies, including roustabout labor, blacksmith and pipe-fitter's tools, evaporating tanks, crystallizing tanks, storage sheds, pumps, freight, etc	5,000
Total	$82,166

CHAPTER XIX.

SELECTION OF PROCESS AND ARRANGEMENT OF PLANT.

WHEN a new copper district has been developed to the point of regular production, it usually becomes necessary to decide what method shall be adopted for the treatment of its ores.

Aside from the very rare conditions where it would be advisable to employ a wet method of extraction—such as a highly siliceous ore, carrying a low percentage of copper and no gold or silver, and of which there are, at present, no examples of moment in operation in North America—we may, in the United States, be confronted with any one, or more, of three great groups of ores, each of which requires distinct and separate treatment, and each of which is characterized by its own peculiar difficulties and advantages, and its own losses and economies.

These are:

1. Native copper ores, or those in which the copper occurs in metallic particles. Example: Lake Superior.

2. Oxidized ores, in which the copper occurs as carbonates, oxides, or silicates, little or no sulphur being present. Example: The Longfellow, Globe, and Bisbee districts of Arizona.

3. Sulphide ores. Example: Pretty much all the important copper districts of the world, excepting those just mentioned.

1. *Native Copper Mines.*—This division is represented mainly by the Lake Superior district. The treatment of the ore consists of a process of mechanical concentration, the resulting concentrates, locally termed "Mineral," being rich and pure enough to require only a simple melting-down and toughening in the refining furnace, combined with the treatment of the resulting slags in cupolas.

The interest of the Lake Superior method centers in the process of mechanical concentration, which, though highly developed, and equipped with admirable machinery, still leaves much to be wished for in regard to losses.

2. *Oxidized Ores.*—While most of the copper veins and deposits of California, Nevada, Utah, Colorado, Montana, Wyoming, and New Mexico contain oxidized ore near the surface, continuing often to a depth of one, or more, hundred feet, it is only in the great carboniferous limestone formation of Arizona that the original pyritous contents of the veins have become so thoroughly and deeply oxidized as to give rise to extensive, and, comparatively, permanent, enterprises based exclusively on these altered ores. While in other districts there may be bodies of oxidized ores sufficiently extensive to warrant the erection of furnaces for their treatment, the metallurgist should ever bear in mind the probability, and, in acid rocks, almost the certainty, of their changing into sulphides at very short notice.

I need say but little about the treatment of these ores, for as the greater includes the less, so does the treatment of sulphide ores and the general arrangement of the plant therefore include the simpler task of smelting the oxide ores.

In inaugurating a plant for treating these simple ores, it must always be borne in mind that where they mostly abound, labor, and especially coke, is very high. Hence the importance of using every possible means to diminish these two expensive items. That every reasonable device to lessen the amount of hand labor required will be adopted, may be assumed as a matter of course; but sufficient care is not always taken to arrange for a self-fluxing charge, this vital point being often left to be considered more carefully after the smelter is completed. The use of a single pound of barren limestone, or iron ore, as a flux, can only be excused by the direst necessity. There are very few practices about a furnace that entail such a lengthy and costly train of evils as the habitual and routine use of barren flux. The following are but a few of its most obvious disadvantages:

(*a*) The flux costs money. It is not only the actual cost of mining, transporting, handling, crushing, analyzing, charging, and superintending it, but also a variable, and often heavy, investment and constant outgo for acquiring, prospecting, and opening-up quarries, and deposits of iron or limestone, building wagon roads or tramways, overseeing and managing.

(*b*) It takes the place of just so much profitable ore, thus reducing the capacity of the works accordingly. A plant that can smelt 200 tons of material daily, and requires $100,000 to erect, will thus have cost $500 per ton of material treated, per 24 hours.

If it is necessary to use 20 per cent. of flux, the plant will have cost $625 per ton of ore smelted, and $20,000, or one-fifth of the entire investment, will have been used for arranging to smelt worthless rock.

(c) It increases the quantity of slag produced, and thus augments the proportionate loss of valuable metals, as it carries no metal itself, and the metal that is contained in 160 tons of ore has got to furnish that lost in the slag made from 200 tons of material, instead of only from its own proper 160 tons.

(d) It increases the cost of labor, fuel, etc., as based on the production.

(e) It encourages a great waste in a direction that is seldom considered, but that is very real in practice. The mere name of "flux" possesses a charm for the blast-furnace foreman, and if given free scope in this direction, he usually develops a passion for its employment that is second in strength only to his desire to use "extra" coke whenever anything is going wrong, or the furnace is not running quite up to its highest standard. The flux bin is thus constantly drawn on for trifling causes, and a comparison at the end of the campaign between the flux consumed, as reported on the furnace-sheet, and the actual amount delivered, according to the quarry foreman, is likely to show a startling discrepancy.

The smelters of oxidized ore like to make tolerably basic slags, because they smelt easily and thus aid the furnace in keeping up a large tonnage. Again, a fusible slag means quick running and a feeble reducing action, and gives a pure black copper, almost free from iron, which is always agreeable. But this very high-grade end-product may be too dearly bought. A more acid slag and a more powerful reduction in the furnace may lessen the losses of copper in the slag enough to counterbalance the slight evils due to a certain amount of reduced iron in the black copper, while a considerable quantity of profitable ore may be substituted for the basic flux.

So long as the great outcrops of cupriferous hematite lasted, this matter did not force itself upon our notice so strongly, but where barren iron ore has to be added to a charge, and in mines where great bodies of cupriferous kaolin exist, it will certainly repay the metallurgist to make a careful study of the various types of slags that consist largely of silicates of alumina and lime, quite possibly calling in the aid of the hot blast.

New enterprises that are starting furnaces in Arizona, or other carbonate districts, should always remember to employ furnace-men that are familiar with the smelting of this particular class of ores. The most skillful matte-cupola smelter often experiences much trouble in handling oxidized ores for the first time, owing to the tendency of metallic copper to chill in the hearth, and the different management required, both for furnace and forehearth.

3. *Sulphide Ores.*—A full discussion and comparison of the choice of methods for the treatment of sulphureted ores of copper would mean an exhaustive analysis of all that is contained in the foregoing volume. I can only sketch in a bare outline, referring the reader to the respective subjects for all details.

As matters stand in American copper-practice to-day, we can simplify our problem by assuming that we are, in any ordinary case, bound to use converters for the production of our black copper (with subsequent electrolytic separation, if gold and silver are present, or probably with immediate refining to ingot, if they are absent), and that, consequently, we must produce a matte approximating 50 per cent. of copper. A higher grade of matte would cause too rich slags, as well as other embarrassments in the smelting process, and a much poorer matte would greatly increase the costs of converting.*

Hence, in selecting our method of treatment, we have only to consider the best means for producing a 50 per cent. matte from the ores at our disposal. Here we are at once confronted with a possible variety of conditions so great as to forbid their detailed consideration in these pages.

For practical purposes, we may divide the ores, that we may be called upon to treat, into three great classes:

1. Where the ore consists very largely of gangue, the same having a much lower specific gravity than the sulphides which it contains. This gangue usually consists of quartz, slate, limestone, feldspar, decomposed granite, diorite, etc. As examples may be cited the veins of Butte, with their granitic gangue, and the beautiful purple ores of Harvey Hill, Quebec, with a limestone gangue, most of the copper ores of Cornwall, and the ores of the Coxheath mines in Cape Breton. Also the African ores of the Cape Copper Company, Ltd., and much of the Quebrada ore.

2. Where the ores contain just the proper amount of silica to

* It is probable that we may soon find it practicable and economical to bessemerize poorer mattes, as is done in Europe.

flux the bases present, including the ferrous oxide produced from the sulphurets. Familiar examples of such favorable ores are the cupriferous and nickeliferous pyrrhotites, with diorite gangue, of Sudbury, Ontario, and the copper-bearing pyrrhotite of the Ely belt in Vermont, with a gangue of quartz, hornblende, and potash mica.

3. Where the copper-bearing minerals are more or less intimately mixed with other minerals of high specific gravity, usually iron pyrites. As examples may be quoted the heavy pyritic ores of Sherbrooke, Quebec; some of the cupriferous Leadville pyrites; the ores of the Rammelsberg, Rio Tinto, and Mt. Lyell deposits.*

For the ores of the first class, the process of mechanical concentration with water presents itself at once, and is, indeed, largely adopted. The Butte mines form the great American center of this industry, and the amount of copper that has been lost there during the past 15 years, in slimes and tailings, is very large. Even with the most approved apparatus and careful running it is usually impracticable to concentrate sulphide ores of copper with a loss of much less than 25 per cent., and as the copper glance and bornite of the Butte district are very rich and have a strong tendency toward the formation of slimes, it can well be supposed that no unusually favorable results can be expected. I do not want to get into hot water by attempting to estimate the present losses in the Butte concentrators, but I may take the liberty of stating that when managing works there some 10 years ago, I found it impossible to keep the concentration losses down to 33 per cent. when estimated on the actual ore that went into the mill and the concentrates that were delivered to the furnaces. Estimates based on the assays of tailings and slimes may be made to give almost any desired results. Since those days, the Butte practice has constantly improved, but the demand for ever-increased production, and the consequent necessity of forcing great quantities of material through the concentration mill, prevents as thorough work as would otherwise be done.

In starting a concentrator on new ores, it is rarely safe to figure on a loss of less than 30 per cent.; not that we *cannot* do closer

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* I am not considering wet processes in this volume; otherwise I should refer to the Rio Tinto leaching process, (as employed in Spain) as a most promising outlook for the future treatment of certain of our low grade pyrites.

work than this, but that in the expensive districts in which most of our mining is done, it will not pay to do it.

In view of the great cheapening of calcining and smelting expenses that the last five years has witnessed, the question will often arise whether it will not be more profitable to do completely away with the concentrator, and smelt the entire ore. This will at once bring up the consideration as to what it will cost to flux the large amount of siliceous gangue present. Under older conditions, where we were too inclined to think that we must have a slag made to order, containing somewhere between 28 per cent. and 32 per cent. silica and 70 per cent. of bases, largely ferrous oxide and lime, any suggestion to attempt to smelt, on an immense scale, an ore carrying an overwhelming proportion of siliceous gangue rock, would have been ridiculous, unless there were an ample store of profitable basic ores available, a luxury that the metallurgist can seldom expect.

At present, however, and especially in the light of the results obtained in pyritic smelting, we may seriously consider the feasibility of producing bisilicate and even trisilicate slags containing from 45 per cent. to 60 per cent. silica. By thus substituting a furnace process for the wet concentration, we may often save two units of copper on a 7 per cent. ore, representing a value of some \$2.40, to which may be added at least \$0.60 for the saved cost of concentration, and perhaps \$1 for the cost of smelting the concentrates (as less than three tons are put into one in ordinary Butte concentration practice). This would give us some \$4 per ton of original ore, out of which must come the cost of smelting it, and the losses in the operation. These losses will be larger than might at first appear; for, although these acid slags will be very clean, they will yet be two or three times as much in weight as would the slags resulting from the same ore, if reduced to concentrates. So that if we make three tons of slag assaying one-fourth of one per cent. of copper, we shall lose as much value as though we only made one ton assaying three-fourths of one per cent.

I am not yet seriously advocating this practice for the great bulk of the Butte ores (though I am quite certain that it would pay the large mines to reduce the grade of the material that is sent to the concentrator, by a more careful selection of all ore fit to go to the furnaces), as it is a question that can be much more accurately solved by the very competent men in charge of the various works there; but it may be an idea worth considering, especially if Whitehall, or any of the promising districts with which Butte is encircled, could render pyritic smelting economical, by furnishing large supplies of the massive, and highly auriferous pyrite, that is so marked a feature in certain of their mines.

Where the ores are rather low-grade and self-fluxing, or where the smelter is so located that a sufficiency of profitable fluxing ores can be obtained, the all-absorbing question of the slag-forming elements falls away, and the metallurgist can devote his attention to the next point: "How can we most economically oxidize the sulphur and iron of our ore, so that the sulphur can pass away as sulphurous acid gas, and the iron be oxidized, and combine with the silica to form a slag?"

As already intimated in a former chapter, there are four feasible methods by which this can be accomplished:

1. By roasting the ore in lump form in heaps or stalls, and smelting in blast, or possibly, in reverberatory furnaces.
2. By crushing the ore and roasting it in automatic calciners, using reverberatories for its subsequent fusion.
3. By pyritic smelting, without any preliminary roasting.
4. By smelting the ore raw, without producing much oxidizing effect, and producing a low-grade matte for bessemerizing. (Not to be thought of until our converter practice for the lower grades of matte becomes at least as good as it is in Europe.)

In weighing the *pros* and *cons* of these methods, there is much more to be considered than the mere metallurgical processes. The business aspect of the case may be of, at least, equal importance. If the surrounding district contains extensive agricultural interests, or is more or less densely populated, and, especially, if the interests of the inhabitants are not so entirely identical with the mining and smelting element as to make them exceedingly tolerant of sulphurous acid gas, heap roasting is out of the question. By a beneficent law of Nature, nearly every individual who joins a mining community proceeds at once to acquire an interest in one or more mines or mills, and thus views the disagreeable concomitants of the metallurgical processes with more leniency than he would show if such were not the case.

Again, it must be remembered that heap roasting will permanently tie up, for each ton of ore smelted daily, as many tons of ore as it takes days to roast it, plus a certain surplus quantity for safety. Thus, if it takes four months, or 120 days, to roast the ore, and we are smelting 200 tons daily, we shall need in constant process of roasting, 120 times 200, or 24,000 tons of ore. To be on the safe side, we shall need 25 per cent. more than this, or about 30,000 tons of ore, in the roast heaps. This represents a considerable capital, though the value of the ore should only be charged up at what it actually cost to mine and put in the heaps.

A wet or windy climate will also interfere seriously with heap roasting, as there is always a considerable proportion of soluble copper in the roasted ore, and the losses, even with reasonable care, may easily amount to 25 per cent. of the entire copper contents. The evil results of strong and variable winds are not so noticeable as those of a wet climate, but are perhaps just as important and real. Aside from a serious loss from the blowing away of the light, pulverulent material so largely formed in roasting ores, the wind causes an irregular combustion in the heaps, that renders quiet, thorough oxidation impossible. About the time the foreman has got his heaps properly covered with fines to withstand a violent current of air, it falls calm, and the amount of oxygen supplied to the interior of the heap is lessened by one-half, or more. Time, quiet, and regularity are the three main essentials to good heap roasting.

Many of the objections to heap roasting are mitigated by the employment of stalls. Wind and weather do much less harm in the compact battery of stalls with paved yard, than amidst the widespread slovenliness of roast heaps. The amount of ore tied up also will not be more than one-eighth as much. The sulphur smoke instead of being diffused through the lower atmosphere, is tolerably well concentrated at one or two points, and discharged through tall stacks.

The cost of roasting in stalls will not usually exceed that of heap roasting, $0.50 per ton being a fair estimate for Western conditions. The results are, on most ores, even more satisfactory than can be obtained in heaps, as the amount of unroasted covering is much less, owing to the enclosing walls. But good results cannot be obtained where the stalls are crowded beyond their proper (and always very small), capacity, and it brings only discredit to the metallurgist and loss to the company, to attempt to help out an insufficient roast plant by running the stalls more rapidly than is compatible with the best possible work.

The first cost of a battery of stalls is considerable. Including the stacks, the grading of the ground, etc., it will seldom be less than $60 for each ton of ore that they are capable of roasting per

24 hours; or $12,000 for a battery to roast 200 tons of ore daily. It may easily amount to very much more than this figure. The repairs and renewals of stalls are, also, a constant and considerable expense.

In common with heap roasting, however, this method offers the great and, often, indispensable advantage of treating and delivering its ore in lump form, to a considerable extent, and is thus peculiarly suited to blast-furnace work, or to new enterprises where want of time and means forbids the installation of an extensive crushing, and calcining plant. Hence, stall roasting is even sometimes employed for preparing lump ores for reverberatory smelting, and furnishes, on the whole, a better product for this purpose than does the kindred process of heap roasting. This results from the fact that the very slow and thorough oxidation that is undergone by the ore in well-conducted heap roasting gives rise to a considerable proportion of ferric oxide, which is peculiarly disadvantageous in the reverberatory process, and which is much more sparingly produced in the rapid operation of stall roasting. Unless the lumps contain numerous fine particles of valuable mineral, enclosed in a massive, infusible, and non-decrepitating gangue rock, the stall-roasted ore will not require further crushing to fit it for the reverberatories.

In planning a plant of stalls, the most important point is to arrange for the cheap and convenient delivery of the ore to the stalls, and thence into the bins, or hoppers, of the smelting furnaces. As the walls of stalls always tend to become racked and distorted by the constant heating and cooling, and especially by the penetration of crystallizing roast products into their joints, it is important to construct them of heavy slabs of some material. Thus there are few joints, and the slabs are too heavy to be easily displaced. While certain kinds of sandstone, slate, etc., fulfill these requirements very well, they are often expensive to quarry, dress, and transport, and, though seemingly capable of standing any ordinary heat, will, after a few roastings, begin to flake and crumble. Blocks of slag, or large slag-bricks, are perhaps the cheapest and most suitable material that can be found for this purpose, and in erecting a new plant distant from other smelters, it is sometimes worth while to delay the building of the stalls, and heap-roast a few thousand tons of ore, which, when smelted, will yield the requisite slag-bricks for the stalls.

If the ore tends to produce an under proportion of fines in the mine, and in the breaking for the roasting process, or yields a very large amount of pulverulent material in the roasting operation itself, it is better to resort at once to reverberatory smelting furnaces and thus escape the costly evils that attend the smelting of too large a proportion of fines in the blast-furnace, and the laborious catching and resmelting of the flue-dust. At the same time, if the necessary capital be forthcoming, it will probably be advantageous to crush all the ore fine and roast it in automatic calciners. Considering the amount of handling that is saved, ore can now be roasted as cheaply in automatic furnaces as in heaps or stalls, and when the losses of values that occur in the cruder methods are reckoned, and the great saving in time and plant resulting from the use of hot calcines in the reverberatory smelters, is considered, the furnace calcination will usually be found the more profitable. Every metallurgist is familiar with the numerous summings-up by authors as to the comparative advantages and disadvantages of the reverberatory and blast-furnace methods of smelting, and, until within the past few years, the résumés were, on the whole, valid and useful. But they must now be set aside and disregarded. Since the late developments in both classes of furnaces, the relations to each other of reverberatory and blast-furnace work have been completely altered, and I cannot to-day venture to make any general statement as to any advantages that the one possesses over the other, either in first cost of construction for a given capacity, economy of running, or ease of management when doing its best possible work. Each case must be judged upon its own suitability for the particular task in view, and speaking comprehensively, I can only say that the blast furnace requires coked fuel and a charge of a certain degree of coarseness, while raw fuel and a fine charge are particularly suited to the reverberatory. For re-treating slag, the cupola is decidedly superior, but for smelting ores, it is doubtful if the blast furnace will save any higher percentage of either copper, gold, or silver than the reverberatory.

It is only when we desire to bring about a strong oxidizing action, and thus produce a richer matte than the chemical composition of the charge properly warrants, that we are forced to turn to the blast-furnace, and then we are beginning to trench upon the domain of "pyritic smelting." On the other hand, when over 40 per cent. of our charge consists of fines (I do not include in this term such concentrates as are coarse enough to be discharged above the jig-sieve), our only economical refuge is the reverberatory.*

There is no object in my discussing pyritic smelting in connection with these other strictly practical subjects; for true pyritic smelting is nowhere as yet practised in the United States on any ores; and it has never been used at all with the purpose of making a 40 per cent. to 50 per cent. copper matte, and a worthless slag. At Tacoma, Deadwood, Leadville, and a few other points, blast-furnaces are being run with a low ore-column and more wind than is necessary for the proper combustion of the coke, and are thus oxidizing a portion of the sulphur and iron of the ore, and, occasionally, obtaining a certain useful amount of heat therefrom, as well as a decided gain in the grade of the matte. But none of them are even claiming to do genuine pyritic smelting, unless it be the Bimetallic Smelter at Leadville, Colorado, which, when I saw it running last summer, was, perhaps, furthest of all from practising this process. The large percentage of coke employed, and the rapid fusion of a heavy pyritic ore forbade any considerable oxidizing effect, and the rate of concentration was astonishingly small.

Nor can I waste space in re-discussing a method which I propose, and have already referred to in a former chapter in detail, and which seems to me to promise very economical results where cheap power is available, and when certain mechanical difficulties have been surmounted. I refer to the simple fusion (without oxidation) of ores, like those of Butte, containing a large proportion of gangue, in large, boshed cupolas resembling iron furnaces, using a very hot blast, and producing a highly siliceous slag, all of which can be made into building and paving bricks of the most valuable quality. The large proportion of sulphur present will prevent the reduction of any metallic iron, the slag being, in fact, mainly a silicate of alumina, lime, and magnesia, with a comparatively low fusion point. The resulting matte will be low grade, and will run directly from the reverberatory forehearths of the blast-furnaces into the Bessemer converters, where it will be blown up to blister copper in two operations, the first one being continuous.

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* I do not for a moment mean to imply that many cupolas are not running successfully on a far larger proportion of fines than that just stated. I have myself run a charge for a long period of time, containing some 80 per cent. of material that would pass a 16-mesh screen. But, except under unusual conditions, I think it would always be more economical to smelt such a charge in reverberatories.

In this way, the entire plant would consist of but two departments; *i.e.*, blast furnaces, and converters. All crushing and roasting would fall away. The raw ore would be charged automatically into the great blast-furnaces, and the escaping gases would be used to heat the blast, as at Mansfeld. It will be seen that this plan aims to throw all the work possible upon the water-power which is supposed to be at our disposal. This would furnish the blast for cupolas and converters, and the only considerable item of expense would be the coke used in the first smelting. The matte is purposely made low grade, because it can be thus produced at the smallest expense, and as the matte, when once made, will be brought up to blister simply by the wind furnished by the water-power, which is the cheapest agent at our disposal, it follows that the lower we make our matte at the outset, and the more we thus save on the costly end of the process, the more the work that we shall be able to throw upon the cheap end of the process, and the less our copper will cost us.

But this scheme is impossible until we have conquered the mechanical difficulties of bessemerizing low-grade matte continuously, and making the converter linings stand. The slag from the first converter operation (concentrating), would be quite poor enough to throw away. As this would remove most of the iron of the matte, the amount of slag resulting from the second bessemerizing operation would be very small, and could be best treated in a separate cupola with the first-class ore, producing a high-grade matte.

CHOICE OF LOCATION AND SITE FOR SMELTER.

The locating and choice of site for a new smelter is an affair of such importance that it demands the very best consideration and judgment on the part of those entrusted with the decision. A great mistake is often made, and many conveniences sacrificed, by trying to get the works too close to the mines that are to supply them with ore. A few miles, more or less, of track between mine and smelter is a matter of but slight importance compared with the grade of the railway. About all the heavy freight has to come from the mine to the smelter, and 100 yards of a bad up-grade on the way to the latter is an infinitely greater obstacle than a mile, or more, of smooth track on a down-grade. While the engineer is making two or three runs to try to surmount the steep pitch, or is taking up his train piecemeal, he could have accomplished the mile of down-grade, and with infinitely less wear on drawbars, locomotive, and rails, and much less expenditure of fuel.

A 3 per cent. grade (158.4 feet to the mile) is pretty bad, if of any considerable length, while a long 5 per cent. grade (264 feet to the mile), is prohibitory to a reasonable train. An engine that will haul 900 tons on a level can only draw 66 tons, at the same speed, on a 3 per cent. grade. In a climate subject to much rainy or foggy weather, rails that run near roast heaps may become so covered with a slimy coating (resulting from the action of sulphuric acid on the distillation products of the roast-wood), that it acts as though the track had been greased, and only the liberal and constant use of sand will neutralize the trouble.

Wherever a heavy train has to make a start, as from the ore bins at the mine, it is well to have the track laid on a very slight down-grade, say 0.5 per cent. (26.4 feet to the mile). This greatly facilitates the starting, and yet is not steep enough to allow the cars to get away if carelessly left standing without brakes.

Next in importance to the problem of getting the ore to the smelter is that of bringing in the fuel and other large supplies. This is still another argument for having the works a considerable distance below the mines, and thus as little as possible above the grade of the valley railway over which the fuel, etc., must pass. A third advantage of such a location is that water will be cheaper, more abundant, and less affected by cold weather than up nearer the head of the valley.

The character of the actual site of the smelter must, of course, depend mainly upon what Nature has placed at our disposal. Where ample room exists, the amount of grading is not excessive, and good foundations can be obtained, a terraced plant will produce a considerable saving in running expenses, and will be found very much more convenient than one built on the same level. I have tried both systems under a considerable variety of conditions, and I should never think of constructing anything but a terraced smelter where circumstances would permit.

The proposed site should be surveyed with the greatest accuracy and platted on a tolerably large scale, say 40 feet to the inch. Accurate contour lines should be run at vertical distances of about 2½ feet. Not a single permanent building or track should be begun until the whole plant is carefully worked out and every point fixed, and sketched in on the map. In this way the grades and curves of the different railway tracks can be laid out to the best advantage, and arranged, so that without wasting any fall the various products shall descend by gravity from one step to the next.

All rough sketches and calculations should be made on tough brown paper and never destroyed. The buildings, furnaces, etc., may be drawn on tracing cloth direct from the original pencil sketches, and these tracings will constitute the original finished drawings, from which any number of blue prints can be taken, as desired. None of these drawings or sheets should ever be folded, nor even rolled more than can be helped. They should be laid flat in the shallow drawers of a large cabinet constructed for the purpose. Each tracing, blue print, or sheet of any description should be numbered and classified,* and there should be a card catalogue by which it can be found without loss of time, and which is capable of indefinite extension.

The main points to consider in determining the exact distance apart vertically of the various terraces are:†

The main track from the mine runs to the adjacent roast-yard, which lies pretty near the top level of the works, but so that the railroad cars can run over the stalls or heaps, and drop the ore on to them without handling. The roasted ore will also be removed in railroad cars running on sunken tracks close to the roasting apparatus. The cars from this, as well as from all other tracks in the works, can be switched on to the high-level track that runs along over the ore bins of the blast-furnaces. In this way, all raw ore, roasted ore, foul slag, coke, bricked flue-dust, quartz and clay for converter linings, etc., that is needed in the blast-furnaces, or on their level, is delivered into the high-level bins in their rear by self-dumping railroad cars.

Where there is not fall enough, or the grades make it inconvenient to run a locomotive over the high-level smelter bins, an inclined plane with stationary engine should be used, and the cars dragged up to the proper level by a wire rope. An inclined plane is more economical to run, and has a very much greater capacity

* Blue prints can be written on in white, by using a strong aqueous solution of cooking soda, instead of ink.

† As space space forbids a detailed consideration of all classes of works, I will assume, for the sake of illustration, the simple case of a plant consisting of stalls (some distance to one side of the main works), cupola furnces with reverberatory forehearths, and bessemer converters. The pig copper will be shipped to distant refining works for treatment.

than an elevator, and the railroad cars can thus be handled direct at very slight expense.

The slag from the blast-furnaces will be granulated and swept away by a strong current of water running in an underground culvert paved with slag-blocks. This culvert will run along in front of the forehearths of the blast-furnaces, and turning at right angles, after leaving the building, will pass under the various tracks and discharge into the valley. The small amount of foul slag produced at the cupolas can be run over a gangway at one side of the converter-room, and dumped into cars standing on the lowest level, to be at once switched back to the blast-furnace bins. The coal for the reverberatory forehearths reaches them through a chute from the level of the charging floor, the coal being brought into the works on the high-level track. In the same way, the quartz and clay for the converters are dumped into the high tier of bins at one end of the cupola building, and go direct into the quartz, and mixing mills, which stand on the intermediate level, so that when the material is completed, it has only to be dumped through a chute to reach the converter building. The ashes from the fire-boxes of the reverberatory forehearths are dumped into the slag launder and swept away by water.

The matte from these same forehearths is tapped direct into the converters, and the latter's product in pig copper and in slag for resmelting, is loaded in railroad cars on the lower or converter track; the former, to go to an electric refinery in the East; the latter, to be switched up to the ore cupola bins. Except to convey a few hundred pounds of cinders daily from the reverberatory forehearths to the slag culvert, and a ton or two of foul slag from the forehearths to the chute overhanging the lowest track, a wheelbarrow need scarcely be allowed inside the works, nor will there be any shoveling worth mentioning.

Outside the works, and just before the main track divides its upper and lower branches, there will be a large pair of track scales. The number, weight, and destination of every car passing over these scales should be taken by the weighmaster, and a reliable record will thus be obtained of everything that comes into or out of the works, as well as all by-products that are returned to the blast-furnaces or elsewhere.

All tracks and switches should, so far as possible, be completed in advance of the buildings, so that the stone, brick, timber, and especially the machinery, may be brought in on the railway cars and placed in position without unnecessary handling.

By obtaining foundation-plans, in advance, of the manufacturers, the foundations for engines, boilers, crushers, and other heavy machinery can be entirely completed and the mortar well set before the machines arrive.

While good foundations are essential to all buildings, furnaces, and machinery, much money and time are often squandered in trying to make them unnecessarily massive and durable. Where labor and stonework are expensive, no massive rock underpinning need be used for the buildings. A much cheaper, and infinitely more expeditious, plan is to simply dig holes and set a good, sound log on end directly under where each post of the building will come. When all these log posts are in place and firmly held in a perpendicular position by filling around them with rammed stone or slag, their upper ends are sawed off perfectly true, and the sills of the building placed directly upon them. The little space between ground and sill may be filled up at leisure by cheap rubble work, as it has no weight to support. The foundations of engines, furnaces, etc., should consist of solid mason-work, though not extending deeper than is necessary to reach rock, solid hard-pan, or undisturbed gravel or sand. This point is as often reached at four feet from the surface as at forty. Good, uniform sand makes an excellent foundation, even for quite tall stacks, though it is sometimes necessary to excavate an area several times larger than the proposed stack-base, and cover it with a horizontal layer of sound logs, adzed to a level, and with all joints well filled with cement mortar. On this foundation comes the foundation proper, and the weight of the superstructure is thus distributed over a considerable area.

A large block of breton, or concrete, makes a still better platform. I have for many years used the following mixture with excellent results: One part, by measure, of sound, hydraulic cement, and three to four parts of clean, sharp sand, thoroughly mixed while dry, moistened to a very stiff paste mixed with four times its bulk of broken stone, and rammed in thin layers into a box-shaped mold of plank, the surface of each layer being roughened, that the next one may adhere to it. After this concrete has hardened for a few days, it is as good as ordinary stone.

Slag blocks make excellent foundations, and may be made by pouring pots of slag into the excavation, or, where possible, by running it in direct from the furnace. In fact, no one who has not paid particular attention to the subject can form any idea of what may be done with slag, even when it is quite basic and ferruginous. Excellent bank walls can be made from it by building up a rough 4-inch or 8-inch wall of bats laid in mud, and braced when needful, at the proper distance from the bank, and pouring in slag until the space is filled. On removing the bats, we have a smooth, monolithic wall, which has cooled so slowly that the slag is usually well tempered and seldom crumbles badly. By a system of sand ridges, well known to cupola men, an entire floor, or yard, may be thus paved almost without expense, and an ingenious outside foreman, with an ample supply of slag at his disposal, will almost dispense with stone quarries.

In most of our Western metallurgical districts, buildings with timber frames covered with corrugated sheet-iron are, on the whole, the most economical. There need be no danger of fire in such buildings, especially if the timbers are kept well covered with fire-proof paint. It is the boarding and light material of a building that causes danger from fire, and not the heavy timbers. Where lumber is cheap, the timber frame is very economical and can be prepared and set up rapidly. Except for the sills and plates, and for a few of the main posts, no tenons or mortices are necessary. The timbers are simply butted together, being slightly notched to prevent slipping, and then strapped or obliquely bolted.

Corrugated sheet iron, No. 24 Birmingham wire gauge, should weigh 1.2 pounds per square foot. No. 22 weighs about 1.5 pounds. The sheets are usually 27½ inches by 96 inches, and are lapped 2½ inches at the sides and 4 inches at the ends, and should extend 6 inches over the eaves. It requires about one-seventh more area of sheets than the surface to be covered.

This corrugated iron naturally makes a very strong arch, and by using iron as heavy as Nos. 13 to 16, arched roofs up to a 30-foot span may be thus constructed, having no support except continuous skewbacks of light angle iron, frequent tie-rods, and a few thin rods to keep the latter from sagging.

Where it is used in smelting works, the life of iron siding and roofing depends mainly upon the integrity of its protecting coating of mineral paint. Dr. Iles, of Denver, has pointed out that silica-graphite paint forms an admirable substance for this purpose, as well as for painting wood-work.

The putting in place and riveting of the corrugated iron should be entrusted only to men who are skilled in this particular work; and, indeed, it is a universal rule, both in building and running works, that experts should be employed at each separate task and properly paid. Only wealthy companies can afford to violate this rule.

Where sulphurous gases are very abundant, and especially in moist climates, the corrosion of sheet iron is too rapid, and we are forced to use either lumber, brick, or stone for our buildings, and to roof them with shingles, slates, or some one of the various gravel and asphalt devices that have answered so well of late years.

By soaking the boards and shingles in a solution of ferrous sulphate (copperas liquor), and subsequently covering them thoroughly with silica-graphite paint, they can be made pretty nearly fireproof. This copperas liquor may be obtained by concentrating the drainage from the roast-heap yard, first precipitating the copper with scrap iron. The best results are obtained by using perfectly green boards, the copperas liquor rapidly displacing the sap of the wood.

In cold climates, all water and steam-pipes should be carefully protected and provided with drip-cocks at all low points. Water pipes of especial importance, and that are peculiarly exposed to freezing, may be safeguarded by running an inch pipe close to them through the trench or box in which they are enclosed, and letting the exhaust steam from some engine or pump constantly pass that way in cold weather. It must be remembered that any dampness in the vicinity of pipes, or in the filling which protects them, greatly increases the conducting power of the material, and consequently its liability to freeze.

The transportation of material about the works and furnaces under conditions where cars are not suitable, may sometimes be very economically effected by mechanical conveyers. Howorth's pan-conveyer for molten slag is an example of such an application of this system of transportation, and cold substances may be cheaply moved, and carried up, or down, quite steep inclinations, by conveyer belts. An overhead, grooved trolley, running on a single rail and provided with differential chain blocks, forms a very cheap and convenient arrangement for transporting pigs of matte, or even quite heavy rocks or castings.

Whether the water supply enters the works from an elevated ditch or pipe-line, or whether it is obtained from wells or streams below the smelter, it is absolutely essential to safety and convenience to always have a large supply, at a powerful pressure, available for fire purposes, feeding jackets and boilers, sprinkling roofs and yards, granulating slag, etc. This should be distributed to all portions of the buildings and yards by means of convenient hydrants, always ready for immediate use. The source of this pressure may either be a reservoir or large tank on the hill above the works, or a suitable standpipe. In either case, the water will probably have to be raised to the required elevation by means of steam pumps. These may be obtained of several reliable and experienced firms, and to suit almost any conceivable combination of circumstances, and it is much safer, unless one has a large practical experience in such matters, to send a full and exact account of the local conditions to the manuacturers, and allow them to select the size and style of pumps. Two points should be borne in mind in this correspondence with the pumpmakers: one is, to allow amply for the maximum work that the pump will ever be called upon to do; for, like a man, a pump is seldom at its best, when called on suddenly for extraordinary duty. The other precaution is to learn who is the practical man in the office of the manufacturer of whom you intend to procure the pumps, and see that your letters are addressed direct to him. Otherwise, you are liable to be put off with advice that comes from men who have not had the necessary practical experience in pump installations.

It would be, of course, unnecessary and wasteful to pump all the water that is used in the works up to the high reservoir. Any portion of the process that requires a constant supply of water should not have this water pumped any higher than is necessary to obtain the desired pressure. To avoid too great a multiplication of pumps, the main pipe through which the water is pumped to the high reservoir may be regarded as a standpipe, and the required amount of water taken from it laterally at such elevations as the case demands. It will be necessary to use a check-valve, opening outward, at each lateral branch.

Neglect to use sufficiently large pipes, and to introduce proper check valves and foot-valves, are among the most frequent causes of waste of power in pumping.

The Riedler system of relieving the weight on the valves adds greatly to the economy and durability of pumps working under considerable pressures.

A pump that has to raise its supply water to some height by suction, should, of course, have a tight foot-valve at the extremity of the suction pipe. But, even when provided with ample air chamber, such a pump is apt to pound and jar in a disagreeable manner. The simplest means of stopping this is by tapping a very small stop-cock into the main suction pipe, close to the pump. This little pipe is provided with a check-valve, opening inward, and at each stroke of the piston the pump sucks a minute quantity of air through it, that effectually prevents the hammering.*

By a simple automatic device, the speed of the pumps may be made to adapt itself to the varying pressure in the pipes. This arrangement, though always useful, is not so absolutely indispensable where there is a large distributing reservoir at a considerable elevation above the works. But where the entire hydrant system depends upon the pumps for its constant pressure, it is necessary to have pumps that are large enough to work up to the maximum demand that can ever be made upon the hydrants. Therefore, during the greater portion of the time they will run very slowly, and the device referred to automatically regulates their speed, so as to keep up a constant and equable pressure in the entire system.

Good fire-hose, wound on proper reels or carriages, and carefully protected from dirt and wet, must be available. The couplings and branches, or nozzles, should receive particular attention, and a frequent fire drill should be instituted, as well to inspect and test the apparatus, as to familarize the men with their respective stations in case of an alarm of fire.

As electricity is used in almost all smelters, at least for lighting, it is an excellent plan to provide each portion of the plant with one or more push-buttons for use in case of fire. When one of these is touched, it rings a gong in the office, and also in the pump-room, and an annunciator in the office indicates the building from which the alarm has proceeded.

Where artesian wells are used, and the supply of water is scanty, the flow can usually be considerably increased by the use of a small air compressor and Pohle pump. The compressed air being introduced into the water-pipe at the bottom of the well, produces a powerful upward current that greatly augments the flow of water.

Fire-clay is expensive at most smelting works and is only needed

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* I am indebted to Dr. Iles, of the Globe Smelter, Denver, for this, and various other little devices.

where very high temperatures are to be withstood. Common (adobe) mud mixed with a little horse manure, the finely-divided straw in the latter acting as a binding material, makes an excellent mortar for lining forehearths and metal-launders, and for general patching.

Where the draft is poor or the coal bad, or too short-flamed, a closed ash-pit and light blast under the grate will usually relieve the difficulty. A small steam jet is frequently used to suck in air, on the principle of the injector. A steam-jet under the grate also serves to keep loose and friable the clinkers that result from coal containing a high percentage of fusible ash. The effect is very striking.

Slack coal, and other inferior coals, can generally be used to best advantage on a step grate, taking care to give it ample area. Such coals may be fed automatically from a hopper, into which the railway cars dump direct.

"*Supplies*" are a great drain on all metallurgical enterprises, and are often consumed in most unnecessary profusion. To keep this item down to the lowest practicable point, and yet to always have an abundant stock of every required article on hand, it is necessary to know exactly what is consumed from day to day, and for what purpose it is used. This can only be done by having a proper storehouse and a thoroughly competent storekeeper. Everything entering the storehouse is charged to him, and not so much as a screw or bolt should be removed from his possession without a written order, showing for what purpose the article is to be used. The storekeeper sends to the office every night a list of such articles as he finds he is getting short of.

If the metallurgist in charge of works intends making any radical change in the manner of running his furnaces, such as greatly increasing the size of his charge, insisting that nine charges per 24 hours shall be smelted, instead of eight, or some such innovation that is peculiarly offensive to all furnace-men, and especially to reverberatory men, and has a good practical knowledge of furnace work himself, his best chance of success is to discharge his old gang if they profess that the new plan is impossible, and put on a gang of strangers who know nothing about furnace work. It is easier to run a furnace on a novel plan with men who know nothing about it than with those who know too much.

It is difficult to avoid cliques about the furnaces, and for this reason a judicious mixture of nationalities will often prevent the deceptions, and the attitude of passive resistance to all improvements, that characterize a body of experienced furnace-men of any one nationality. A mixture of Irish and Cornish furnace-men, with an American foreman, usually works well, as the men all dislike and distrust each other so much that they find it impossible to combine against the common enemy.

Mechanical devices, however, are the great ally of the employer, and when we once get rid of the process of skimming, reverberatory work will have lost much of the atmosphere of mystery which has always surrounded it, and the old race of prejudiced and bigoted furnace-men will gradually become extinct.

In the meantime, much can be accomplished by appealing to the interests of the men, and instituting contract work, under certain limitations. This should only be attempted, however, by very experienced superintendents. It is also essential to guarantee the men that they shall enjoy the advantages arising from an increased output, for a considerable time in advance. Else they will assume that the contract has been given them in order that their employer can discover how much they are capable of doing when they work their hardest, and that if they show their hand by exerting themselves and making a handsome thing out of it, it will soon be taken away from them and they will be expected to accomplish as much by day's work as they showed they were capable of doing by contract.

Although one must not always expect to be met in the same spirit, the only way in which large numbers of workmen can be successfully managed is by employing extreme patience and a little more than perfect fairness, recollecting that it is easier for the educated, than for the ignorant, man to see the justice there may be in his opponent's claims, or the weak points that may exist in his own.

THE END.

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